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[" The Cyanide Process of Gold



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THE CYANIDE PROCESS

OF

GOLD EXTRACTION:

*A TEXT-BOOK FOR THE USE OF MINING STUDENTS,
METALLURGISTS, AND CYANIDE OPERATORS.*

BY

JAMES PARK,

PROFESSOR OF MINING AND DIRECTOR OF OTAGO UNIVERSITY SCHOOL OF MINES;
FELLOW OF THE GEOLOGICAL SOCIETY OF LONDON; MEMBER OF THE AMERICAN
INSTITUTE OF MINING ENGINEERS; MEMBER OF THE INSTITUTE OF
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PREFACE TO THIRD EDITION.

THE favourable reception accorded to earlier editions has encouraged me to revise the old matter, and at the same time add much new material, which, for the most part, relates to lead-smelting of gold-slimes, the treatment of sulpho-telluride ores, and filter-press practice. In Western Australia, the adoption of filter-pressing was mainly determined by a combination of peculiar local conditions, namely, the natural tendency of the gold ores and matrix to form slimes, the scarcity of fresh water, and the saline character of the only water available for milling purposes. The density of the brackish waters rendered the settlement of the finer material in the decantation process so slow, laborious, and imperfect as to make the use of filter-presses almost imperative for quick and effective treatment. The slimes, it should be noted, formed the first or primary product of the mills, and were of high value. In South Africa, where the question of filter-pressing versus decantation has been engaging attention for some time, the slimes are in all cases a secondary product and of low-grade value.

Filter-pressing has been adopted with much success by the Waihi Company, in New Zealand, for the treatment of ordinary slimes, but, on the other hand, it should be mentioned that several neighbouring companies are satisfied with the decantation process for the treatment of similar material. It is quite clear that the relative merits of the two processes must, in every case, be determined by exhaustive trials extending over a period of, say, four to

six months, so as to eliminate the element of fortuitous chance in favour of either, and thereby enable a reliable estimate of costs to be prepared. With South African and New Zealand conditions, the solution of the problem is obviously one of cost. In Western Australia, the question of cost is subordinate to that of expediency.

The introduction of lead-smelting of gold-slimes marks a notable advance in cyanide practice, and if the claims of Mr. Tavenor, the author of the process, are verified by more comprehensive trials—as indeed seems most probable—lead-smelting will in a short time displace the old smelting and acid-treatment processes, at any rate in all the larger permanent cyanide plants.

There is still much room for special chemical investigation in several directions, and in this respect the researches of a number of accomplished metallurgical chemists in South Africa have added much valuable material to the literature of the cyanide process. The successful regeneration of foul cyanide solutions is still unsolved, and is at present the subject of investigation by several American chemists. The results of their work will be awaited with much interest.

I have to acknowledge my indebtedness to Mr E. G. Banks, M.I.M.M., and Mr Frank B. Allen, M.A., B.Sc., for special contributions on Waihi slime and filter-press practice and Kalgoorlie sulpho-telluride practice respectively; and to the proprietors of the *Mining Journal*, *Mining and Engineering Journal*, and *Australian Mining Standard* for permission to make extracts from articles which appeared in their columns at different times. In these and all other cases, due acknowledgment and reference are made in the text.

JAMES PARK.

UNIVERSITY, DUNEDIN, N.Z.,
September 1903.

PREFACE TO THE FIRST ENGLISH EDITION.

THE favourable reception accorded to the Third Edition of this work, published in New Zealand, has enabled the author to again revise and enlarge the text, with a view of placing before his readers, earlier than was anticipated, the latest information available on this progressive branch of metallurgy. The general plan of the original work, which was intended for the use of mining students, metallurgists, and cyanide operators, has been retained in the present edition, which is the first published in England, and into which only such changes have been introduced as the author considered needful. Of late years the application of scientific investigations and methods to the treatment of ores has rendered metallurgy more and more dependent on chemical knowledge, and in no department is this more obvious than in the Cyanide Process of Gold Extraction, which often presents the most perplexing problems, due, in many cases, to the simplest causes.

In the present edition several new illustrations and tables have been added, while the information relating to the treatment of slimes, the analyses of solutions, and cyanide poisoning, has been greatly extended; while, by the adoption of slightly smaller type, and closer setting, the actual number of pages has been reduced, although the text has been considerably enlarged.

Since the revision of these pages, wet-crushing and cyanide treatment have largely superseded dry-crushing and direct

cyaniding in New Zealand, and in every case their installation has been attended with complete success, notwithstanding the large proportion of slimes at some of the mines. The ores are mostly chalcedonic in character, and contain both coarse and fine gold associated with silver sulphide. The silver occurs in constantly varying proportions, requiring varying strengths of solution to obtain adequate extractions, and unremitting care on the part of the metallurgist in charge.

The results obtained in the treatment of these comparatively complex ores are of world-wide interest, and have been embodied by the author in a separate appendix.

JAMES PARK.

AUCKLAND, *January* 1900.

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THE
CYANIDE PROCESS
FOR THE
EXTRACTION OF GOLD AND SILVER.

CHAPTER I.

THE M^CARTHUR-FORREST PROCESS.

It has long been known that gold and silver are soluble in solutions of alkaline cyanides, but it is only within the past few years that this knowledge has been applied on a commercial scale to the extraction of the precious metals from their ores.

The discovery of the fact that the dilute solution of potassium cyanide is a solvent for natural gold, ranks among the most remarkable discoveries of the present century in metallurgical science; and the widespread and successful application of the fact must mark an epoch in the history of gold extraction for all time.

SCOPE OF THE PROCESS.

The cyanide process can be applied with success to the treatment of free-milling ores in which the gold occurs in fine particles, or of tailings and concentrates resulting from wet-crushing and copper-plate amalgamation, or dry-crushing and pan-amalgamation. It can also be used for the treatment of many so-called refractory ores, especially those in which the gold occurs in such a finely-divided form that even amalgamation in pans fails to

recover a satisfactory percentage of the values; or of ores in which the gold is coated with a film of metallic oxide or sulphide, rendering it non-amalgamable, and ores in which the gold is associated with, or entangled in, a highly pyritic matrix.

All the common ores of silver are more or less soluble in dilute solutions of cyanide. Those most readily soluble are the chloride (AgCl) and the sub-sulphide (Ag_2S), and these are fortunately the most abundant; but the rate of dissolution of silver and its ores is much slower than that of gold, and is accompanied by a higher consumption of cyanide.

LIMITATIONS OF THE PROCESS.

The cyanide process cannot be applied with success to the treatment of ores in which the gold occurs even in a fairly coarse condition. When an ore contains a proportion of both fine and coarse gold the cyanide process may be used to extract the fine gold, but a supplementary treatment will have to be used to recover the coarse gold, since the slowness of the dissolution would take too long for a commercial basis of working.

With free-milling ores of the latter class the recovery of the coarse gold is generally effected by copper-plate amalgamation; and, in the case of wet-crushing, this treatment precedes the cyanide leaching, while, in the case of dry-crushing, it follows it.

The experience gained during the use of the cyanide process has shown that solutions of potassium cyanide, even when very dilute, act most energetically on all the sulphide, oxide, and carbonate ores of copper, and also on the sulphides of antimony and bismuth; hence, when any of these is present, even in small proportion, the treatment of the ore becomes difficult, and sometimes impossible, on account of the great consumption of cyanide. In practice it is found that an unduly large consumption of cyanide is generally accompanied by a low rate of extraction of the gold and silver contained in the ore or tailings.

From the foregoing it is obvious that the process will be most successful in the treatment of ores in which the gold occurs in a very fine state, and in which the quantity of base minerals or metallic salts, destructive to cyanide, is small.

Further, the author ascertained as the result of many simultaneous working trials in the N.Z. Government Metallurgical Works that argentiferous gold ores which were amenable to treatment by the Washoe pan-amalgamation process, in most cases yielded better results by cyanide treatment, even when they contained a small percentage of zinc and lead.

An intelligent knowledge of first principles backed by experience of working details and working requirements has led to many ingenious adaptations, the tendency of which has been to greatly widen the scope of the original cyanide process. A notable case in point is the successful treatment of the rich sulpho-telluride ores of America and Kalgoorlie, which affords satisfactory evidence of the progressive trend of this important branch of metallurgy.

CHAPTER II.

THE CHEMISTRY OF THE PROCESS.

WHEN gold is acted on by an aqueous solution of potassium cyanide, a solution is obtained which, when evaporated, yields octahedral crystals having the composition of the auro-potassic cyanide (AuKCy_2), which is a double cyanide of gold and potassium.

The exact reaction which takes place when gold is dissolved by potassium cyanide is not yet well understood, being still a subject of much doubt and uncertainty. According to some authorities, the gold is oxidized before it is dissolved; while others maintain that the cyanide is first oxidized and then acts on the gold.

The reaction suggested by Elsner in 1842 is the one now most generally accepted by chemists. It is represented by the following equation:—



According to the above equation, an ounce of oxygen is required for every pound of potassium cyanide employed for the dissolution of the gold. This view has received substantial support from the author's experiments in 1891, and from those of Skey* in 1892, and has since been proved experimentally correct by Maclaurin † in his classical paper on the subject

The valuable researches of Skey and Maclaurin have shown that the rate of dissolution of pure gold, under theoretical conditions, reaches a maximum in passing from dilute to concentrated solutions of potassium cyanide. By actual experiment, it was proved that the maximum rate was reached with a 0.25 per cent. solution of cyanide. On a working scale the maximum varies with the character of the mineral constituents of the ore, and can easily be determined by a series of laboratory experiments.

A weak solution is always more active than a strong one, and

* Skey, *N.Z. Mines Report*, 1894.

† *Jour. Chem. Soc.*, May 1893, p. 724.

Maclaurin considers that this remarkable fact may be accounted for by supposing that the rate of dissolution of gold is partly dependent on the number of cyanide molecules in a unit volume; and partly on the number of oxygen molecules in the same volume. One of the most important results of his exhaustive experiments was the demonstration of the fact that the solubility of oxygen in cyanide solutions decreases with concentration of the solution.

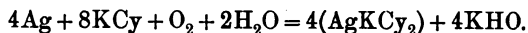
Weak aqueous solutions of cyanide exert a very marked action on gold and silver when these metals are associated with ores of copper and antimony. This circumstance becomes very prominent during the treatment of cupriferous ores on a large scale.

The cyanides of the alkaline metals are soluble in water, while those of the heavy metals, with the exception of gold and mercury, are insoluble. The insoluble salts are, however, soluble in excess of potassium cyanide.

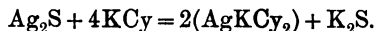
The use of an oxidizing agent that will readily part with a portion of its oxygen in a cyanide solution forms the essential feature of several new patent cyanide processes. The employment of such an agent serves to accelerate the dissolution of the gold contained in the ore. The artificial aeration of cyanide solutions is undertaken to supply atmospheric oxygen with the same object.

Consumption of Cyanide.—According to Elsner's equation, about 4·5 lbs. of cyanide should dissolve 100 ounces of gold, but in practice it is found that it takes nearly forty times that quantity. The causes which operate in the practice of the process to effect so large a consumption of cyanide, over that required by Elsner's simple equation, are at present not fully investigated.

To dissolve 100 oz. of silver would require 7·5 lbs. of cyanide, according to the equation :—



For the dissolution of 100 oz. of silver existing as the sub-sulphide (Ag_2S), 7·01 lbs. of cyanide would be required by the following equation :—



The potassium sulphide resulting from the dissolution of silver sulphide also tends to cause a further loss of cyanide by precipitating gold which will require an excess of free cyanide to redissolve it. It is the need for this excess of cyanide which necessitates the use of comparatively strong solutions in the treatment of argentiferous gold ores.

Potassium cyanide is, chemically, a most active organic compound, possessing the property of forming so large a number of complicated and unexpected combinations in the presence of mineral acids and base metals, that its reactions and behaviour with different classes of ore, and under varying conditions, can only be unravelled by much patient research, both in the laboratory and under working conditions.

During the treatment of ores by the cyanide process, the most puzzling difficulties are continually met with, requiring the constant care and attention of the metallurgist in charge.

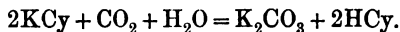
Causes of Loss of Cyanide.—Some of the principal and more obvious causes of the enormous loss of cyanide which takes place in the working of the process are as follows :—

- 1. Loss by absorption in wooden vats or tanks.
- 2. Loss by decomposition by atmospheric carbon dioxide.
3. Mechanical loss in residues, and by dilution of solutions during washing.
- 4. Loss by decomposition due to the presence of mineral acids and salts.
5. Loss due to presence of ores soluble in cyanide.
6. Loss when gold exists as amalgam.
7. Loss due to the presence of charcoal in kiln-dried ore.

Loss by Absorption in Vats.—This is especially noticeable in new plants. At the Witwatersrand Goldfields, the loss from this cause is said by Mr. C. Butters to amount to a pound of cyanide per ton of tailings treated. At the first monthly "clean up" in a new plant, the actual extraction is often twenty or more per cent. below the theoretical, but after a few months it generally rises to within three to six per cent. of the extraction as determined by assay.

With iron or steel vats there is no appreciable loss by absorption.

Loss due to Decomposition by Atmospheric Carbon Dioxide.—The carbonic acid gas of the atmosphere decomposes potassium cyanide with the formation of potassium carbonate, and the liberation of hydrocyanic (prussic) acid, thus :—



The prussic acid thus liberated would be neutralized by any caustic alkali present in the cyanide solution.

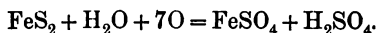
Mechanical Loss in Residues, and by Dilution during Washing.—During washing there is an inability to extract the whole of the cyanide from the residual tailings. The dilution of the cyanide solutions also occasions a loss of cyanide in washing. A large quantity of dilute cyanide solution is formed, a portion only of which can be utilized to make up fresh solutions.

Loss by Decomposition due to Mineral Acids and Salts.—The metallic minerals most commonly found associated with gold in quartz veins are iron pyrites, copper pyrites, zinc blende, galena, and antimonite. By far the most common and abundant of these is iron pyrites.

It has been shown by Skey and others that clean fresh iron pyrites is not acted on by working solutions of cyanide. The decomposition products of this mineral, however, act most destructively on cyanide, and the obvious conclusion to be drawn from this is that the treatment of pyritic tailings, or concentrates, by cyanide should be undertaken with as little delay as possible; more especially when the pyrites occurs in the marcasite form, which is much more prone to oxidation than the cubical or isometric form.

In the shallow parts of mines, the pyrites is generally oxidized to ferric oxide, which does not act chemically on cyanide, but causes a mechanical loss through the formation, both in wet and dry crushing, of extremely fine slimes, which are very absorbent and retentive of cyanide solutions.

Iron pyrites (FeS_2) is decomposed by atmospheric oxygen in the presence of moisture into the soluble ferrous sulphate and free sulphuric acid, according to the following equation:—



In the kiln-drying of ores to be dry-crushed, the heat to which the ore, often in large pieces, is subjected, is not very uniform, especially in large kilns. With pyritic ores the sulphides are decomposed at certain temperatures into oxides and soluble sulphates; and at higher temperatures the latter salts are converted into oxides.

The steam generated from the moisture in the fuel and in the ore itself assists these reactions. In the kiln, where the temperature is high, reducing gases are evolved, and these may impede the oxidation of the sulphide, causing the formation of lower sulphides and basic sulphates, which are insoluble in water, but react on cyanide.

For these reasons, and also for economy, ores intended for dry treatment by cyanide should never be kiln-dried. In the case of developed mines, with an assured and steady output, there is nothing to justify the practice, but much to condemn it.

As we have seen, the atmospheric oxidation of pyrites results in the production of free sulphuric acid and ferrous sulphate. This ferrous sulphate may in turn be decomposed, by the action of the air, into insoluble basic sulphates. Thus, partly oxidized pyritic ores or tailings may contain free sulphuric acid, soluble ferrous

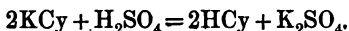
sulphate, insoluble basic sulphates, and probably also traces of other basic salts of complex and variable composition, all of which react upon solutions of potassium cyanide, thereby causing a loss of cyanide.

The reactions which are most likely to take place in acid ores or tailings in the presence of cyanide are :—

(a.) The liberation of hydrocyanic acid.

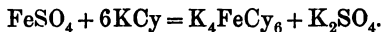
(b.) The formation of ferro- and ferri-cyanides.

The free acids in the ore react on the cyanide as shown by the equation :—

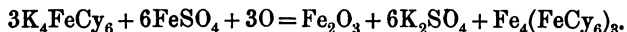


Feldtmann considers it possible for the hydrocyanic acid thus liberated to diffuse itself through the ore and dissolve appreciable quantities of gold.* For this reason he strongly condemns the practice of washing acid tailings in the leaching vats, as these must always contain a residual portion of cyanide from which prussic acid would be liberated. Any gold dissolved by this gas would be carried away in the water or alkaline wash; and to avoid this possible source of loss, which he thinks may account for the mysterious discrepancy sometimes found between the assay and the actual extraction, he recommends the system of washing in one vat and leaching in another. On the other hand, Skey, when discussing this subject with the author, stated that hydrocyanic acid was not a solvent for gold. It is obvious, however, that this liberated HCy in the presence of any residual alkali in the vat, would form an alkaline cyanide capable of dissolving and removing gold, and Feldtmann probably had this combination in his mind at the time of writing.

Of the iron salts, the one of most common occurrence in pyritic ores or tailings is the soluble ferrous sulphate (FeSO_4), which reacts with potassium cyanide to form potassium ferro-cyanide and sulphate, thus :—



The potassium ferro-cyanide thus formed is, in its turn, reacted on by any excess of ferrous sulphate still present with the production of Prussian blue according to the equation :—



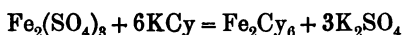
A blue colour in the solution, on the surface of the tailings, or in the seams of the staves of the vats, indicates a large consump-

* Feldtmann, *Notes on Gold Extraction*, p. 5.

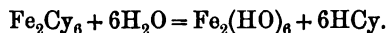
tion and loss of cyanide due to imperfect washing and neutralization of the acidity in the preliminary treatment.

A white scum or precipitate is sometimes seen on the surface of the solutions when they are coming off acid. This precipitate turns into Prussian blue by exposure to the air and light.

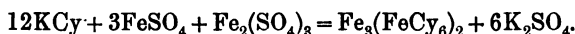
The normal ferric sulphate $\text{Fe}_2(\text{SO}_4)_3$ is insoluble in water, and cannot be removed by ordinary water-washing. It reacts with potassium cyanide, causing a loss of cyanogen due to the liberation of prussic acid and the formation of the ferric hydrate, as shown by the two following equations :—



and



It is probable that in most partially oxidized pyritic ores and tailings the ferrous and ferric sulphates exist together, the former in large excess. In this case the decomposition of the cyanide would result in the production of ferrous cyanide and potassium sulphate, thus :—



In the case of earthy pyritic ores, the weathering or oxidation of the metallic sulphides would result in the production of sulphates of magnesia, lime, or alumina. The action of these sulphates is not very clear, but they most likely react on cyanide with the liberation of prussic acid, accompanied by the formation of the hydrated oxide of the metals in question, which would be precipitated as an insoluble incrustation in the solution pipes.

The above reactions clearly emphasize the necessity of a most careful preliminary alkaline treatment of pyritic material, in order to avoid undue loss of cyanide, and ensure satisfactory results.

All the iron salts and earthy sulphates can be rendered innocuous by the application of an alkali before treatment with the cyanide. By this means all the soluble iron salts are precipitated as ferrous hydrate, which rapidly oxidizes to ferric hydrate; while the basic ones soon oxidize in the presence of the alkali. It is important to remember that the alkali should be applied before, and not with the cyanide solutions, as these iron salts will destroy the cyanide as much in a strongly alkaline as in a nearly neutral solution. When the tailings contain free acid only, alkali and cyanide should be applied together.

Loss due to Presence of Ores Soluble in Cyanide.—The sulphide, oxide, and carbonate ores of copper, and the sulphides of antimony and bismuth, are acted on by potassium

cyanide both in weak and strong solutions, and thereby cause a loss of cyanide in proportion to their abundance in the ore. In the treatment of an ore containing as little as 0.25 per cent. of copper the consumption of cyanide will be doubled.

It is during the treatment of cupriferous ores that the selective action of weak cyanide solutions becomes most apparent. An ore may contain sufficient copper to decompose a 1 per cent. solution of cyanide and give a low extraction of gold, whereas a 0.35 per cent. solution would dissolve proportionately less copper, and give a fairly satisfactory extraction of the gold. But the same results would be obtained even in the absence of copper, for it has already been shown that the rate of dissolution of gold reaches a maximum in passing from dilute to strong solutions. Hence a 0.35 per cent. solution should extract more gold than a 1 per cent. solution, the weaker solution being nearer the strength at which a maximum rate of dissolution occurs, and which has been proved experimentally to be a 0.25 per cent. on pure gold.

The cyanide treatment of ores, and zinc-precipitation of the gold, have shown the existence of copper in ores in which no trace of that metal could be detected, even by the most rigid chemical examination on large samples. An instance of this came under the notice of the author, at the Crown mines at Karangahake. The ore being treated there consisted of almost pure white quartz, free from all metallic impurities; nevertheless, a portion of the zinc in the precipitation boxes was often coated with a film of bright metallic copper. The copper could not be derived from an outside source, or from any of the mechanical fittings in the mill or cyanide plant, and Mr. James Napier, the metallurgist and chemist in charge, was of the opinion that it existed in the ore in an infinitesimally small quantity, and only became manifest on the zinc turnings after the treatment of hundreds of tons of ore.

Copper pyrites is oxidized to the soluble sulphate at low temperatures, and this salt requires a greater heat to decompose it than iron pyrites. It is, therefore, probable that a portion, at least, of this mineral present in an ore, being dried in kilns, preparatory to dry-crushing and direct cyanide treatment, would be sulphatized, and thereby cause an appreciable loss of cyanide in a manner similar to that caused by the decomposition products of iron pyrites.

Malachite and azurite, the green and blue carbonates of copper, are both readily soluble in dilute solutions of cyanide, with the production of copper-potassic cyanide and liberation of prussic acid.

Antimonite, the grey sesqui-sulphide of antimony, is also readily

acted on by weak cyanide solutions. It is frequently met with in the gold-bearing oars of the Thames and Reefton goldfields. The presence of a small percentage of antimonite in the large accumulation of tailings at Boatman's Creek, near Reefton, is said to have caused all attempts to treat them to end in failure, chiefly owing to the large consumption of cyanide and the low rate of extraction.

Loss of Cyanide when Gold Exists as Amalgam.—It is well known to most millmen that a considerable portion of the gold in tailings, resulting from copper-plate amalgamation or pan-amalgamation, exists in the form of amalgam. When such tailings have to be treated the cyanide has to dissolve the mercury as well as the gold, thus causing a larger consumption of the solvent than would be necessary if the gold existed in a free state.

According to Gmelin, mercury is not dissolved or acted on by potassium cyanide; but the practical working of the cyanide process has shown that his conclusion is contrary to actual experience.

At the cyanide works of the Cassel Gold Extracting Company, at Waihi, where a large stack of tailings and residues from pan-amalgamation were treated, 75 lbs. of mercury were collected in the condenser attached to the furnace for roasting the zinc slimes. The mercury thus recovered was only a small proportion of the mercury dissolved by the cyanide, and afterwards precipitated in the zinc-extractor with the bullion. On every occasion when the roasting of the zinc slimes was being conducted, so much mercury was volatilized that the vapours pervaded every part of the buildings, condensing on every cool surface, and amalgamating all objects of gold and silver worn by the workman.

The volatilization of mercury during the roasting of zinc slimes, resulting from the treatment of tailings, was noted by the author on several occasions at the Government Experimental Cyanide Works, and is of frequent occurrence at the cyanide plants at Kuaotunu. The same circumstance was noted by Dr. Scheidel at the Sylvia Cyanide Works at the Thames, where mercury was found in the zinc-bullion in considerable quantities.

The mercury generally occurs in the tailings in the form of amalgam in a very fine state of subdivision, and is dissolved by the cyanide, together with the associated gold and silver. It is precipitated with the bullion in the zinc precipitation boxes. When the zinc slimes are oxidized the greater portion of the mercury is volatilized.

Loss of Cyanide due to the Presence of Charcoal in the Ore.—It has long been known to chemists that charcoal

cyanide will necessarily occur; but with these sulphides, in addition to loss of cyanide, there will be a loss of gold, and a still greater loss of silver, in proportion to the quantities present. This loss is brought about by the sulphur—that is, the alkaline sulphide—sulphurizing these metals to form sulphides with them,* the sulphide film so formed upon the metal preventing, or greatly retarding, the proper action of the cyanide solution.

That gold does combine, and very readily, with the sulphur of both the alkaline sulphide and of hydrogen sulphide, Skey has already shown.†

It is to precipitate the sulphur that gets into the cyanide in the cyanide process that Mr. McArthur has proposed to use, or does use (as per patent), a soluble lead salt dissolved in the cyanide.

The problem for the chemist at the cyanide works is to find a practical method, whereby all the sulphur of antimonial and cuprous sulphides can be made to combine with the cyanogen, rather than with the potassium of the cyanide.

The following results obtained by Skey show how extremely objectionable alkaline sulphides are, when present in the cyanide solution.

A rather strong solution of the cyanide, containing a small proportion of sulphur, was placed over a strip of gold coupled with a piece of copper-glance (sulphide of copper), but no solution of gold was perceived; however, on substituting chalcopyrites for the copper-glance, the gold was rapidly removed.

This experiment shows that the gold was sulphurized at the outset by the alkaline sulphide present in the cyanide, and that it required connecting with a substance of a strongly negative kind in order to effect the decomposition of the auriferous sulphide so formed.

Further experiments of a different kind showed that while pure 1 per cent. cyanide solution dissolved a given weight of gold in ten minutes, a solution of the salt of the same strength, but containing $\frac{1}{100000}$ part of sulphur (as a sulphide),‡ required two hours to dissolve the same weight of gold. The speeds were as 12 to 1 in favour of the pure cyanide.

The following results show to what extent even a gentle sulphurizing, or flouring of the gold, interferes with its solution:—

Gold sulphurized 60 seconds in K_2S , dissolved in cyanide in 62 minutes.

* *Trans. N.Z. Inst.*, vol. iii, p. 216.

† *Trans. N.Z. Inst.*, vol. xxi., 1888.

‡ *Trans. N.Z. Inst.*, vol. xxi., 1888, "On the Preparation of Artificial Chromes."

Gold sulphurized 54 seconds in K_2S , dissolved in cyanide in 50 minutes.

Gold sulphurized 1 second in K_2S dissolved in cyanide in 36 minutes.

Gold, clean, dissolved in cyanide in 12 minutes.

The gold was well washed from adherent potassic sulphide before being placed in the cyanide. Making clean gold the unit, the approximate times of dissolution are 1 : 3 : 4 : 5.

Action of Sulpho-cyanides.—It has been held by some metallurgists that the presence of these in working solutions of cyanide is injurious, retarding the dissolution of the gold. As the result of much research, Godfrey Doveton, formerly of Camp Bird Mills, Ouray, Colorado, informs me that he has found that a solution containing potassium sulpho-cyanide up to a certain point was *more active* than a solution of corresponding strength in KCy, with sulpho-cyanide present, and that even 2.50 grams. of KCyS in 100 c.c. of solution did not influence the extraction unfavourably. The salt alone, in solution in water, is a slow solvent for gold.

Experiments on mill solutions show much the same result, and he has come to the conclusion that the presence of even considerable quantities of sulpho salt in working solutions should not cause any uneasiness.

The Action of Manganese Oxides on Cyanide.—During the treatment of a parcel of ore from the Komata gold-mine, near Waitekauri, the author found there was an unusual consumption of cyanide. The ore consisted of soft mullocky, friable quartz, coloured quite black by a large percentage of pyrolusite and wad, and containing a trace of nickel and cobalt.

A series of experiments were afterwards made to determine the cause of the loss, and the results of these at first led the author to the conclusion that the manganese oxides oxidized a portion of the cyanide to cyanate. It is well known that pyrolusite parts with a portion of its oxygen under the influence of heat alone, but more readily so in the presence of an easily oxidizable substance. Further research showed, however, that the loss was due to the cobalt in the ore, which dissolved somewhat readily, thus consuming cyanide. It is interesting to note that the dissolved cobalt was precipitated with the gold and silver on the zinc in the extractor box, and, like copper, was found to interfere with the precipitation of the gold.

The Action of Oxygen-bearing Agents.—It seems probable that in ores containing copper or other base metal soluble in a solution of potassium cyanide, the base metal would, from its preponderance in the ore, necessarily utilize the greater amount of

the available oxygen, thereby tending to render the dissolution of the gold slow and imperfect. Hence it is reasonable to conclude that the employment of an oxidizing agent that would supply the deficiency of oxygen in such complex gold containing ores would be beneficial. That the use of an oxygen-bearing agent is justified in special cases, seems well established by the experience of the author and the results reported by reliable metallurgical chemists in South Africa, America and Western Australia.

CHAPTER III.

LABORATORY EXPERIMENTS.

THE cyanide process is essentially a chemical one, and a commodious and well-equipped laboratory forms one of the most important and necessary parts of the whole plant.

It is the duty of the metallurgist in charge to determine, by actual experiment, the lowest strength of cyanide solution required to extract an adequate percentage of the gold, and also to devise means of overcoming the problems which are inseparable from the treatment of different classes and grades of ore, with so active a compound as potassium cyanide.

The daily output of ore from a mine is subject to continual change, both as regards physical condition and chemical composition, hence the treatment requires, within certain limits, corresponding modifications, to obtain the maximum extraction at the minimum cost. To obtain these results, the metallurgist must be a trained analytical chemist, full of resource and originality.

The testing and valuing of the ores before, during, and after treatment must be entrusted to a careful and trustworthy assayer. The testing and making up of the working solutions are very simple operations, which may be left to experienced and intelligent foremen who possess a knowledge of arithmetic extending as far as decimals.

With free-milling gold ores the actual working extraction will generally be as high as that obtained in the laboratory, but, in the case of ores containing copper or antimony, too much reliance must not be placed on the laboratory experiments.

The author's experience is that high extraction may be obtained in the laboratory from ores totally unsuited for treatment by the cyanide process on a working scale.

The conditions on the one hand are theoretical, on the other actual, and before adopting the cyanide treatment for a sulphide or other ore, working experiments should be made on parcels ranging from two to five tons, in order to ascertain the consumption of

cyanide and actual extraction. If the working trials are successful the cyanide treatment may be adopted with confidence.

On the other hand, in the case of an ore containing comparatively coarse gold, the laboratory experiments—where the sample is *hand-crushed*—will give lower results than those obtained in practice in the cyanide works. The author made a number of experiments on an ore from Marlborough, N.Z. The average extraction in the laboratory was under 40 per cent., while the cyanide plant extracted over 60 per cent. At the battery the ore was dry-crushed through a 60-mesh screen, and investigation showed that a large portion of the gold was reduced fine enough to pass through the screen, and thus became amenable to cyanide treatment.

THE ACTUAL EXPERIMENT.

1. Procure six bell-jars, about four and a half inches in diameter. When bell-jars are not procurable, lamp glasses or clear glass pint bottles, with the bottoms cut off, will answer the purpose quite well. In the neck of each jar fit a cork, perforated with one hole. Through the hole pass a short length of glass tube, on the end of which place a few inches of pliable black rubber tubing. On the end of the rubber tubing place a screw-clip, by means of which the rate of percolation of the cyanide solutions can be regulated to a nicety.

2. Now invert the jars, and fix them in a wooden frame, so as to stand upright. In each jar place a thin layer of small rounded pebbles, about the size of French beans; above the pebbles place an inch of coarse sand, and above this, half-an-inch of fine sand. Above the fine sand place a piece of loose scrim, the diameter of the jar. This completes the filter-bed.

When a large number of cyanide experiments are being made, a box divided into three compartments, to hold the three grades of material for the filter-bed, should be kept well replenished and near at hand.

3. Next procure a fair sample of the pulverized ore to be tested, weighing, say, six or eight pounds. Mix thoroughly, and carefully assay to accurately determine the original value.

Check assays should always be made, and if there is a serious discrepancy between the assay and its check, amounting to over 3 per cent. of the value, fresh assays should be made. The assays form the basis of the calculations and final results of the experiments, and hence the greatest accuracy should be aimed at.

When the ore to be tested is from the battery, or mill, it should be placed in a jar in the condition that it comes from the mill,

except, of course, when the tests are to determine the degree of fineness, which would give the best economic extraction.

When the ore is hand-pulverized, a separate portion should be reduced to pass through, say, a 30-mesh, 40-mesh, and 60-mesh sieve respectively. Separate tests should be made of each grade, so as to determine to what extent the extraction is affected by the varying fineness of the ore.

4. Introduce into each jar 10 or 12 ounces of the powdered and sampled ore, the value of which has been obtained by careful assay. Mark the jars 1, 2, 3, 4, 5, and 6.

5. In the case of tailings or ores containing iron pyrites, or other base metallic sulphides, the samples in the jars should be washed once or twice with clean water to remove any soluble sulphates. With very acid tailings a very dilute alkaline wash may be applied. (Test for acidity, see Chapter IV.)

6. No general rule can be laid down as to the strength of the cyanide solutions to be used, as this will depend as much on the character of the sample as on its gold value; but the strength of solution used on a working scale for any class of ore seldom exceeds 0.6 per cent. All the ores of silver, copper, arsenic, and antimony act on and consume cyanide, and when either of these is present a stronger series of solutions will have to be tried than in the case of clean ores. With the latter a useful series of solutions would contain:—

0.05%, 0.1%, 0.15%, 0.2%, 0.25%, 0.3% of cyanide.

With pyritic ores or tailings, or those containing copper, antimony, or arsenic compounds, the most instructive series would be:—

0.15%, 0.20%, 0.25%, 0.3%, 0.35%, 0.4%.

In the event of all the tests being unsatisfactory, it would be advisable to try both stronger and weaker solutions than those already employed according as the character of the ore or material may suggest.

It is necessary with every new ore to make a number of laboratory experiments to ascertain the strength of cyanide solution required to extract an adequate proportion of the gold and silver contents.

7. To each jar, already charged with the ore, add the same weight of solution as of ore. The excess of solution is required because a large portion immediately finds its way into the filter-bed. Record the numbers of the jars and the strength of cyanide used in each. Regulate the screw-clips so that the percolation will take at least thirty hours. A longer time may be tried if the first trials are not successful. With very dilute solutions, or when testing highly pyritic material, it may be necessary to continue

the leaching for six days or even longer before satisfactory results are obtained. If the solution comes through too quickly, return it again and allow it this time to percolate more slowly.

8. When the leaching is complete, wash with two washings of clean water, allowing the wash-water to drain as rapidly as possible. The washing is complete when the wash-water no longer gives an alkaline reaction.

9. Test the strength of the spent solution to ascertain the percentage of cyanide used. The solution and washings are collected and measured together, then tested for cyanide.

The consumption of cyanide can be calculated by determining the strength of the combined solution and washings, and making an allowance for the increase in bulk due to dilution.

On a working scale the consumption of cyanide is generally much less than that shown by the laboratory experiments.

Sometimes the cyanide and different washings are kept separate and evaporated down with the addition of litharge (in the manner described under The Assay of Cyanide Solutions), and the gold actually extracted by each, calculated separately. The results afford an instructive lesson on the value of successive washings.

10. Remove the leached and washed residues from the jars, dry, mix thoroughly and again assay. As the residues will probably be excessively low-grade, it will be necessary to take 1000 or 1200 grains for the assay determination. Then calculate the percentage of extraction from each jar by difference, recording the results and assay values as follow :—

	Original Value.			After Leaching.			Percentage of Recovery.
	ozs.	dwt.	grs.	ozs.	dwt.	grs.	
Gold, .	2	4	12	0	4	12	89.9
Silver, .	1	0	4	0	3	6	83.9
Value, .	<u>£9</u>	<u>0</u>	<u>0</u>	<u>£0</u>	<u>18</u>	<u>4</u>	<u>89.8</u>

The calculation is simply a matter of proportion. As an example take the recovery of gold.

	ozs. dwts. grs.			
Original gold,	2	4	12	per ton.
After leaching,	0	4	12	„
Extracted,	<u>2</u>	<u>0</u>	<u>0</u>	„

Then if two ounces were extracted from 2 ozs. 4 dwt. 12 grs., what would the extraction be from 100?

2 ozs. 4 dwts. 12 grs. : 2 ozs. :: 100 : 89.9 per cent.

$$\frac{2 \times 100}{2.225} = 89.9 \text{ per cent.}$$

An easy and expeditious method of calculating the percentages of extraction in the laboratory-test is to use the weights of the bullion, gold, and silver (in grains or grams) as the basis of computation instead of the same, extended as ounces, dwts., and grains.

Example:—

	Original Assay.	After Leaching.	Extracted.	Recovery per cent.
Bullion, . .	·0020	·0002	·0018	90.0
Gold, . .	·0018	·0001	·0017	94.4
Silver, . .	·0002	·0001	·0001	50.0
Value, . .	<u>£9 2 0</u>	<u>£0 10 3</u>	<u>£8 11 9</u>	<u>94.3</u>

The calculation:—

$$\begin{array}{r} \text{For Bullion.} \\ \cdot 0018 \times 100 \\ \hline \cdot 002 \end{array} = 90$$

$$\begin{array}{r} \text{For Silver.} \\ \cdot 0001 \times 100 \\ \hline \cdot 0002 \end{array} = 50$$

$$\begin{array}{r} \text{For Gold.} \\ \cdot 0017 \times 100 \\ \hline \cdot 0018 \end{array} = 94.4.$$

$$\begin{array}{r} \text{For Value.} \\ 8.587 \times 100 \\ \hline 9.1 \end{array} = 94.3.$$

11. Compare the results obtained and adopt the strength which gives the highest extraction.

REMARKS.—With a series of experiments it will be found that the percentage of extraction, or rate of dissolution of the gold, reaches a maximum with a cyanide solution of a certain strength, and that above and below this strength the rate of extraction rapidly diminishes. The strength of cyanide solution which dissolves the maximum percentage of gold will depend on the character of the ore. The so-called selective action of cyanide is not so apparent in the laboratory experiments as it is in practice.

On a working scale it soon becomes evident in the treatment of base sulphide ores that a strong solution of cyanide of potassium dissolves a large proportion of the base metals and a small proportion of the gold, while a weak solution dissolves a large proportion of the gold and a small proportion of the base metals.

CHAPTER IV.

CONTROL, TESTING, AND ANALYSIS OF SOLUTIONS.

To Test the Strength of Cyanide Solutions.—This is an operation of great simplicity, and can be performed with accuracy and expedition by any intelligent foreman by a volumetric method of estimation. The standard solutions should always be made up under the personal supervision of the chemist in charge of the works.

Three different volumetric methods may be used for the determination, namely :—

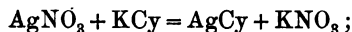
1. By standard solution of silver nitrate.
2. By standard solution of mercuric chloride.
3. By standard solution of iodine.

By Standard Silver Nitrate Solution.

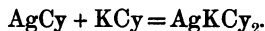
This is the method generally adopted in cyanide plants. It is a modification of Liebig's volumetric estimation of cyanogen.

The reaction depends on the fact that when a solution of silver nitrate is added to a solution of potassium cyanide, the cyanogen unites with the silver, appearing as a white precipitate, which is immediately dissolved by any free KCy, which may still be present, forming a double cyanide of potassium and silver.

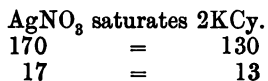
This reaction is shown by the equations :—



and



A standard solution of silver nitrate can be made up from the molecular weights of the constituents as follows :—



With grams, use a decinormal solution ; then if 17 grams of silver nitrate are dissolved in 1000 c.c. of water, 1 c.c. will be equal to .013 gm. of KCy.

TO MAKE STANDARD SILVER NITRATE SOLUTION.—Take 17 grams of silver nitrate (triple crystallized if procurable), and dissolve in one litre (1000 c.c.) of distilled water. In large works, where much testing is going on, it is advisable to dissolve 34 grams in two litres ; then place in stoppered-bottle and mark.

TO TEST SOLUTIONS OF KCy :—

1. Fill a burette with silver nitrate solution.
2. Measure 13 c.c. of cyanide solution to be tested from another burette and transfer to a smaller beaker. To obtain accurate results add a few drops of potassium iodide solution to the beaker and shake.
3. Run in standard AgNO_3 solution cautiously from the burette till the white precipitate formed just *ceases to re-dissolve* when the beaker is shaken ; that is, when a faint permanent opalescence appears the reaction is complete.
4. Read off number of c.c. of standard solution used, and divide by 10. The result will represent the percentage of available KCy. For example :—

Suppose 13 c.c. of KCy sol. took 14.5 c.c. of AgNO_3 , then—

$$\frac{14.5}{10} = 1.45\% \text{ of KCy.}$$

If a strong solution is being tested, in order to save AgNO_3 measure off, say, 3 c.c. or 4 c.c. of cyanide solution instead of 13, and titrate with silver standard. Thus, if 4 c.c. required 6 c.c. of silver nitrate, 13 c.c. would require 19.5 ; and 19.5 divided by 10 = 1.95% KCy.

Even greater accuracy may be obtained in testing strong solutions, such as those in the dissolving tank, together with a saving of silver nitrate, by measuring off 13 c.c. of the strong solution and diluting with water to 130 c.c. Then measure off 13 c.c. of this diluted solution and titrate with silver nitrate as described above. Note the number of c.c. of standard solution required to complete the reaction, and this will represent the percentage of KCy in the strong solution, for since the 13 c.c. of dilute solution contained only a tenth of the original 13 c.c. of strong solution, there is hence no need to divide the quantity of silver nitrate by ten.

To test the strength of very dilute cyanide solutions, measure off 130 c.c. of the solution, titrate with silver nitrate, and divide the number of c.c. of standard required by 100, and the result will give the percentage of available KCy, thus :—

130 c.c. of cyanide solution required 5 c.c. of standard, then—

$$5 \div 100 = 0.05\% \text{ KCy.}$$

To avoid calculation and reduce the liability to make mistakes in reading the burette, a standard solution of silver nitrate can be made up by dissolving 13·07 grams * of silver nitrate in 1000 c.c. of water. To test a cyanide solution with this, measure off 10 c.c. and titrate with silver nitrate. Note the number of c.c. of standard required to complete the reaction, divide by ten, and the result will be the percentage of available KCy, thus :—

If 10 c.c. of cyanide solution required 5 c.c. of silver nitrate,

$$5 \div 10 = 0.5\% \text{ KCy.}$$

REMARKS.—Two burettes should always be used ; one to measure the cyanide solution, and one for the silver nitrate standard. The gram burette should be graduated to 1-10th gram. Erdmann floats should always be used, so as to obtain the exact reading.

The addition of two or three drops of a 2 per cent. solution of potassium iodide to the solution to be tested renders the end reaction more defined, and reduces the danger of over-estimating the KCy due to the alkalinity of the solution.

In an interesting paper on "The Titration and Use of Cyanide Solutions," † Mr. Walter H. Virgoe, Chief Chemist to the Mexican Gold and Silver Recovery Company, shows that where copper is present in the solution, the use of an iodide indicator fulfils the further purpose of indicating the point where the silver nitrate has titrated all the free cyanide, and is about to attack the cyanide in chemical combination with the copper, thereby preventing the over-estimation of the free cyanide. As an example, he says that a solution containing 0.3 per cent. of copper and titrating 0.52 per cent. of cyanide with silver nitrate alone, may titrate only 0.13 per cent. of KCy correctly if KI be used.

Virgoe finds that in titrating pure solutions of potassium cyanide, the amount of indicator used makes no difference whatever, but when copper is present, even in small proportion, he shows that with different amounts of indicator very dissimilar percentages of cyanide are obtained. For this reason he points out the advisability of using a minimum addition of KI before titration of solutions containing copper.

To Test Cyanide Solutions with Grain Standard Solution.

1. When grain burettes are used, make up a solution of silver nitrate by dissolving 170 grains in 10,000 grains of pure water.
2. Measure off, from a burette, 130 grains of cyanide solution to be tested. Add a few drops of KI.

* Thus :—17 : 13·08 :: 13 : 10.

† *Trans. Inst. Min. and Met.*, London, 1901-1902.

3. Fill a burette with the silver nitrate standard.
4. Run silver nitrate into cyanide solution until the white p.p. which at first forms just ceases to re-dissolve. Note the number of grains required.
5. The number of grains of silver nitrate solution used to titrate, divided by 100, will give the percentage of available KCy.
6. For example, if 56 grains of silver nitrate were used, then :—

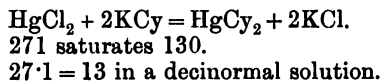
$$\frac{56}{100} = 0.56 \text{ per cent. of KCy in the solution.}$$

By Standard Mercuric Chloride Solution.

(1.) When a solution of mercuric chloride is added to a solution of potassium cyanide, a cyanide of mercury is formed, but is at once dissolved by any excess of KCy present.

When all the free or available KCy has been used, a bluish-white opalescence of HgCy_2 appears if a slight excess of mercuric chloride is added. This permanent opalescence indicates the end of the reaction.

(2.) TO MAKE UP STANDARD SOLUTION.—Use the equation :—



From the above molecular weights, dissolve 27.1 grams of mercuric chloride in 1000 c.c. of distilled water; then 1 c.c. will equal .013 gram. of KCy. Place in a stoppered bottle and mark.

(3.) THE ACTUAL DETERMINATION—

- (a.) Fill a burette with standard mercuric solution.
- (b.) From another burette measure off 13 c.c. of the cyanide solution to be tested, and to this add about 3 c.c. of dilute ammonia.
- (c.) Now run in standard mercuric solution very cautiously, with constant shaking, until a permanent bluish-white opalescence is produced.
- (d.) Note the number of c.c. of standard required to complete the reaction; divide this number by 10, and the result will be the percentage of available KCy present; thus, if—

6.5 c.c. were required to complete titration, then—

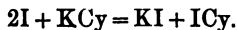
$$6.5 \div 10 = 0.65\% \text{ KCy.}$$

REMARKS.—With pure substances this reaction is very delicate, but with cyanide solutions, containing much impurity, it is not so reliable as the silver nitrate method. Caustic alkalis do not interfere with the reaction. The author has made a number of simultaneous tests, with working cyanide solutions, by the silver nitrate and mercuric chloride methods, and the results obtained were practically the same throughout.

By Standard Iodine Solution.

(1.) This method depends on the fact that when a solution of iodine is added to one of potassium cyanide, the iodine loses its colour so long as any undecomposed cyanide remains.

(2.) **TO MAKE UP STANDARD IODINE SOLUTION.**—Use the reaction :—



254 saturates 65.

25·4 = 6·5 in a decinormal solution.

Therefore, to make a standard solution, weigh out 25·4 grams of iodine, place in a beaker with 200 c.c. of water, and add sufficient potassium iodine to completely dissolve the iodine with frequent shaking.

When the iodine is dissolved, make up to 1000 c.c. with pure water, and place in a stoppered bottle. Then :

1 c.c. = ·0065 grm. KCy.

(3.) **THE ACTUAL DETERMINATION**—

- (a.) Fill a burette with the standard iodine.
- (b.) From another burette measure off 6·5 c.c. of cyanide solution to be tested, and to this add carbonic acid (20 c.c. of ordinary soda water will do) to convert the caustic and mono-carbonate alkalis, contained in all commercial cyanide, into bi-carbonates.
- (c.) Now run in standard iodine, cautiously and slowly, until a slight but permanent yellow colour is produced.
- (d.) Read off the number of c.c. of standard employed, divide by 10, and the result will be the percentage of KCy required.

REMARKS.—This method does not give reliable results in the presence of sulphides, or when the cyanide solution is muddy or discoloured.

To Make up Cyanide Solutions.—There are two different methods of making up solutions in common use in cyanide plants.

In some cases the requisite amount of solid cyanide salt is added to the sump-solution; in others, the working strength is made up by adding strong solution from the dissolving tank to the sump solution.

The following exercises will render these methods clear :—

1 lb. of pure KCy dissolved in 100 lbs. of water gives a 1 per cent. solution; therefore, if you have a vat containing 100 cubic feet of water to make up to, say, 0·6 per cent., you would require 37·35 lbs. of pure KCy.*

Thus $100 \times 62\frac{1}{4} = 6225$ lbs. of water,

and if 100 lbs. of water require 0·6 lbs. KCy, 6225 lbs. would require—

$$100 : 6225 :: 0·6 : x$$

$$\frac{6225 \times 0·6}{100} = 37·35 \text{ lbs.}$$

Commercial cyanide is seldom pure; you would, therefore, have to use a greater quantity to make up the required strength.

Suppose the crude KCy contains 78 per cent. of KCy, then :—

$$78 : 100 :: 34·35 : 7$$

$$\frac{100 \times 37·35}{78} = 48 \text{ lbs. crude KCy.}$$

The same form of calculation will do for making up any required quantity of cyanide solution. Suppose 4 ozs. of a 0·5 per cent. solution were required.

Then if 100 ozs. of water require 0·5 ozs. of cyanide, how much would 4 ozs. require?

$$100 : 4 :: 0·5 : x$$

$$\frac{4 \times 0·5}{100} = 0·02 \times 480 = 9·6 \text{ grains.}$$

If you have a 0·2 per cent. solution and you wish to make it up to, say, 0·5 per cent., subtract the 0·2 per cent. already in the solution from 0·5 per cent., leaving 0·3 per cent. required. Then proceed to make up as directed in the preceding paragraph.

EXERCISES.

- (1.) I have 4000 lbs. of sump solution containing 0·2 per cent.

* See Constants at end of Chapter.

of available KCy, which I wish to make up to a 0·5 per cent. solution, how much additional KCy will be required?

$$100 : 4000 :: 0·3 : x$$

$$\frac{4000 \times 3}{100} = 12 \text{ lbs. pure KCy.}$$

If your crude KCy salt contains only 82 per cent. of KCy, then :—

$$\frac{12 \times 100}{82} = 14·6 \text{ lbs. crude KCy required.}$$

(2.) How many lbs. of solid cyanide salt of 75 per cent. strength should be used to make up 10 tons of a 0·4 per cent. working solution? Ans.—119·46 lbs.

(3.) How many lbs. of solid cyanide salt of 82 per cent. strength should be used to make up 5 tons of a 0·45 per cent. working solution, using a sump solution containing 0·15 per cent. of KCy for making up? Ans.—40·97 lbs.

(4.) How many lbs. of a 14 per cent. stock cyanide solution should be used to make up 10 tons of a 0·4 per cent. working solution, using a 0·18 per cent. sump solution for making up?

Solution.—This is easiest determined by "Alligation," the tea-mixer's rule of proportion, thus :—

$$\begin{array}{rcl} \text{Strong solution } 14·00 & \times & \cdot 40 \text{ working solution} \\ \text{Working solution } \cdot 40 & \times & \cdot 18 \text{ sump solution} \\ \hline & & \end{array}$$

Proportion of weak = 13·60 + ·22 = proportion of strong.
Neglecting decimals, then :—

$$1360 + 22 = 1382.$$

Here we have 1382 parts or lbs. of the required mixture, containing 22 of strong and 1360 of weak solution; therefore, if 22 lbs. of the strong solution give 1382 lbs. of the required mixture, how many lbs. will be required for 10 tons of the mixture?

$$1382 : (10 \times 2240) :: 22 : x$$

$$\frac{22400 \times 22}{1382} = 356·58$$

The answer is, therefore, 356·58 lbs.

(5.) How many lbs. of a 22 per cent. stock solution should be used to make up 9 tons of 0·5 per cent. working solution, using a 0·12 sump solution for making up? Ans.—351·27 lbs.

(6.) How many lbs. of a twelve per cent. stock solution should be used to make up 10 tons of a 0·6 per cent. working solution, using

0.15 per cent. sump solution for making up? Before drawing from the stock solution first utilize 4 tons of a 0.8 solution already in the solution vat. Ans.—359.74 lbs.

Solution of No. 6.—First find out how much of the working solution can be made up from the 4 tons of 0.8 per cent. cyanide solution, thus:—

$$\begin{array}{rcl} \text{Strong solution} & .80 & \times \\ \text{Working solution} & .60 & \times \end{array} \begin{array}{l} .60 \text{ working solution} \\ .15 \text{ sump solution} \end{array}$$

Proportion of weak = .20 + .45 = proportion of strong.

Neglecting decimals, then:—

$$20 + 45 = 65 \text{ of required mixture.}$$

Then, if 45 of the strong (0.8 per cent.) give 65 of the required mixture, the 4 tons already in the solution tank will give 5.77 tons, thus:—

$$\begin{array}{l} 45 : 4 :: 65 : x \\ \frac{4 \times 65}{45} = 5.77. \end{array}$$

And $10 - 5.77 = 4.23$ tons to be made up from the 12 per cent. stock solution, thus:—

$$\begin{array}{rcl} \text{Strong solution} & 12.00 & \times \\ \text{Working solution} & .60 & \times \end{array} \begin{array}{l} .60 \text{ working solution} \\ .15 \text{ sump solution} \end{array}$$

Proportion of weak = 11.40 + .45 = proportion of strong.

Neglecting decimals:—

$$1140 + 45 = 1185 \text{ of mixture.}$$

Now, if 45 lbs. of the stock solution (12 per cent.) give 1185 lbs. of the required mixture, how much will 4.23 tons require?

$$\begin{array}{l} 1185 : (4.23 \times 2240) :: 45 : x \\ \frac{4.23 \times 2240 \times 45}{1185} = 359.74 \text{ lbs.} \end{array}$$

TO DILUTE CYANIDE SOLUTIONS.

(7.) How many tons of a 0.45 per cent. working solution would 6 tons of a 0.8 per cent. solution of cyanide make, using water for dilution? Ans.—10.66 tons.

Solution:—

$$\begin{array}{l} .45 : .80 :: 6 : x \\ \frac{80 \times 6}{45} = 10.66. \end{array}$$

(8.) How many tons of a 0.4 per cent. working solution of cyanide would 8 tons of a 0.6 per cent. solution make, using a 0.12 per cent. sump solution for dilution? **Ans.**—13.71 tons.

Solution :—

$$\begin{array}{rcl}
 \text{Strong solution} & \cdot 60 & \times \\
 \text{Working solution} & \cdot 40 & \times \\
 \hline
 \text{Proportion of weak} & = \frac{\cdot 20}{20} + \frac{\cdot 28}{28} & = \text{proportion of strong} \\
 & & = 48 \text{ of mixture.}
 \end{array}$$

Now, if 28 of the strong solution give 48 of the required mixture, 8 tons will give 13.71 tons, thus :—

$$\begin{array}{l}
 28 : 8 :: 48 : x \\
 \frac{8 \times 48}{28} = 13.71 \text{ tons.}
 \end{array}$$

(9.) How many tons of a 0.6 per cent. cyanide solution would 8 tons of a 0.7 per cent. solution make, using a 0.2 per cent. sump solution for dilution? **Ans.**—10 tons.

To Test the Strength of Crude KCy.—Commercial KCy is formed when any nitrogenous organic bodies, such as hoofs, clippings of hides, wool, and blood are fused with potassium carbonate. This product is very impure, and is lixiviated in a vessel containing finely-divided metallic iron, yielding the yellow prussiate of potassium ($K_4Fe_6Cy_6$), which is the starting-point of all cyanogen compounds.

Crude cyanide of potassium is formed by the action of heat on the yellow prussiate, thus :—



The chief impurities in commercial cyanide are black carbide of iron, alkaline carbonates, and sometimes alkaline chlorides and sulphides in small quantities.

To accurately test the strength of the solid cyanide salt, for the free or available KCy which it contains, proceed as follows :—

- (1.) Break a cake of KCy in two, and select a piece, say, a pound in weight, showing the whole thickness of the cake in section.
- (2.) Reduce this pound to a coarse powder, sample well, and further pulverize to a moderately fine powder.
- (3.) Weigh out 1 gram of powdered and sampled KCy.
- (4.) Dissolve in pure water and make up to 100 c.c.
- (5.) Measure off 13 c.c. of this solution and titrate with silver nitrate standard solution from a burette as previously described. Note number of c.c. of standard required to

form a permanent p.p.; divide by 10 and this will give the amount of KCy in 1 gram of the crude salt.

For example: Suppose 13 c.c. of KCy solution required 7.5 c.c. of standard, then:—

$$\frac{7.5}{10} = .75 \text{ KCy in 1 gram,}$$

which is equal to 75 per cent. of KCy in the crude salt.

THE ASSAY OF CYANIDE SOLUTIONS.

The estimation of the gold contents of cyanide solutions in large works where a great many determinations are made daily must be effected by a method both expeditious and accurate. Several alternative methods are given below, all of which give reliable results with ordinary care. Method I. is in very common use. It has the advantage over methods II. and III. that it can be used for the determination of both gold and silver contents, which is necessary in the case of argentiferous gold ores. Methods II. and III. are silver nitrate processes devised by Andrew F. Crosse, the well known South African metallurgical chemist. Method IV. is a copper sulphate process used by Walter H. Virgoe for some years for the rapid assaying of cyanide solutions. It is recommended by Crosse. Like method I. it can be used for valuation of gold and silver contents.

Method I.—

- (1.) Measure half a pint of solution and evaporate slowly to a small bulk in a round iron drying-dish, over a Bunsen flame, or on the furnace lid. As the evaporation proceeds, rub the sides down so as to collect the whole of the dissolved salts at the bottom.
- (2.) To the solution add 600 grains of litharge. Mix well; evaporate cautiously to dryness.
- (3.) Then transfer to a clay-crucible and mix with 200 grains of glass-powder, 100 grains of soda, and 48 grains of argol (bi-tartrate of potash). Cover with a little borax, and fuse. When fused, pour and allow to cool.
- (4.) Cupel the lead-button and weigh the resulting bead of bullion. If the ore contains silver, part so as to determine the weight of gold and of silver; then refer to the table at the end of the chapter to ascertain the quantity of each per ton of solution.

When the resulting gold is weighed with gram weights, refer to

the Gram Table ; and, when in grains to the Grain Table. (See end of chapter.)

REMARKS.—When a large number of determinations have to be made, ordinary enamelled plates and mugs form efficient evaporating dishes ; in this case also the litharge can be stirred into the solution before the evaporation begins.

At many cyanide works the cyanide solutions are assayed by evaporating a measured portion of the solution in a boat of sheet-lead and then scorifying the residue, and cupelling. When the solutions are charged with base metallic cyanides the results are not generally so reliable as when the solution is evaporated with litharge and afterwards fused in a clay crucible.

Method II. (Crosse).—

(1.) Measure half a pint of cyanide solution, and add silver nitrate solution until a precipitate ceases to form. The silver salt should be added a little at the time, and the solution well shaken after each addition. All the gold in the solution is precipitated as argentic-aurio-cyanide.

(2.) Allow the precipitate to settle ; decant off the clear solution ; filter and dry the precipitate and mix with 200 grains litharge, 100 grains glass-powder, 100 grains soda, and 48 grains of argol. Fuse, pour, and cupel the lead-button.

(3.) Extract the bead of bullion from the cupel, flatten, and part without weighing.

(4.) Weigh the resulting gold and calculate the results.

Method III. (Crosse).*—

(1.) Pour 500 c.c. of cyanide solution into an evaporating dish. Put it in a stink cupboard with a good draught.

(2.) Add nitric acid till the solution shows an acid reaction.

(3.) Boil 15 minutes.

(4.) Then add $\frac{1}{2}$ gram of silver dissolved in silver nitrate.

(5.) Filter ; fuse the filter-paper and contained precipitate as usual with litharge, and flux ; then cupel and weigh resulting bead.

Method IV. (Virgoe).—

(1.) To a litre of solution add excess of weak sulphate of copper solution.

(2.) Acidify with hydrochloric, nitric, or sulphuric acid.

(3.) Filter. The precipitate, which is white and flocculent, contains all the gold and silver. The bluish or greenish colour of the filtrate indicates that excess of copper sulphate has been added.

* *The Journal of the Chemical and Metallurgical Society of S.A.*, May, 1902.

(4.) Wash precipitate, dry and place in a scorifier with metallic lead.

Crosse prefers to fuse the precipitate in a crucible with litharge and to cupel resulting lead button.

Notes on Virgoe's Method.—In a discussion on his method before the Chemical and Metallurgical Society of South Africa, Mr Virgoe supplied the following useful particulars :—

"In dealing with ordinary solutions, I add copper sulphate carefully until no further precipitate forms, stirring the solution in the meanwhile, and taking care to add only a very slight excess of copper sulphate, attested by the faint tinge of blue in the filtrate. Allow the precipitate to stand for a minute or two, H_2SO_4 is then added till the solution is acid, no more, no less. An addition of sodium sulphate produces in the filtrate a slight additional precipitate which is found to contain a very small amount of gold. I find the results are very accurate. For instance, 1 litre of solution was found to contain by careful duplicate evaporation tests, 8.27 milligrams; the copper sulphate method run in duplicate yielded 8.22 and 8.23 milligrams, just a little low, but the test is severe and amply demonstrates to my mind the correctness of the method if properly run.

"As far as I have observed there seems to be a tendency, if strong solutions of copper sulphate are added to strong solutions of cyanide, towards the formation of cupric salts, hence I use weak solutions of copper sulphate, and should expect bad results if the precipitation of gold were attempted by my method from a very strong solution of cyanide with a strong solution of copper sulphate."

ANALYSIS OF CYANIDE SOLUTIONS.

Working solutions of potassium cyanide soon become contaminated with ferro-, ferri-, and sulphy-cyanides and cyanates of the alkaline earths and base metals soluble in KCy, some of which have a beneficial and others an injurious effect. In order, therefore, to be able to modify the treatment to obtain a higher rate of extraction, or to reduce the consumption of cyanide, or effect a better precipitation, it is necessary to know something of the constitution of the working solutions which generally contain, even in the treatment of the purest ore, an assemblage of complicated compounds requiring much technological skill for their estimation.

The analysis of cyanide solutions is an object to which many accomplished chemists, mostly in South Africa, have devoted

much attention. Many useful schemes have been formulated, the object in most cases having been to devise reliable methods, devoid of too much refinement, yet capable of every-day application in cyanide works.

The "Shaking Test" for Consumption of Cyanide.—This method is much used in the laboratories of the Cassel Cyanide Company, and affords a rapid and fairly approximate estimate of the consumption of cyanide with different classes of ore. It is useful for comparative purposes, and as a preliminary means of determining the most suitable strengths of cyanide solutions for laboratory experiments.

(1.) Take 200 grams of the ore and place in a stoppered bottle with, for example, 100 c.c. of a 0·5 per cent. cyanide solution, and shake for twenty minutes.

(2.) Allow contents of bottle to settle; draw off a portion of the clear solution with a pipette and test for KCy. If it contains 0·2 per cent. of KCy, then 0·3 per cent. has been consumed or decomposed.

(3.) When much cyanide is used up, test the ore for acidity by Feldtmann's method given below.

To Test Ores and Tailings for Acidity.—

(1.) Weigh out 224 grams of ore and shake up with 250 c.c. of water in a tall glass-jar or cylinder.

(2.) Fill a burette with a standard solution of soda, and titrate the ore-solution in the jar until the reaction is neutral to test (litmus) paper.

(3.) Every c.c. of the soda solution used will represent 0·1 lb. of caustic soda to be added to every ton of ore (or tailings) in a wash before the cyanide treatment.

To MAKE STANDARD SODA SOLUTION.—Dissolve 10 grams (or 154·3 grains) of caustic soda in 1000 c.c. of pure water, and place in a secure bottle.

REMARKS.—During the titration, the litmus paper should be dipped in clean, pure water from time to time to remove the adhering particles of ore so that the reaction may be clearly seen.

Tests for Alkaline Sulphides in Cyanide.—Alkaline sulphides act injuriously in cyanide solutions during leaching, and it is important to detect their presence. They are all soluble in water.

First Test:—To the clear cyanide solution add a little acid. If an alkaline sulphide be present, sulphur will be liberated, imparting a cloudy appearance to the solution.

Second Test:—In the clear solution place a clean, bright silver coin. It will become black and tarnished if a sulphide be present. This and the preceding test will not detect minute quantities.

Third Test :—The most delicate test is by means of the nitro-prussides. These are formed by adding a little nitric acid to a solution of ferro- or ferri-cyanide of potassium.

Add a few drops of a solution of nitro-prusside to the cyanide solution. If an alkaline sulphide is present, even in minute quantity, the solution will assume a brilliant purple colour.

Fourth Test :—When a dilute solution of a soluble lead salt, such as the acetate, is added to a solution containing an alkaline sulphide, a blackish brown precipitate of lead sulphide soon forms.

When the alkaline sulphide exists in the presence of free cyanide, a white precipitate of lead cyanide and carbonate will immediately form on the addition of lead acetate, thus tending to render the lead sulphide precipitate yellowish or nut brown.

THE ANALYSIS OF CYANIDE SOLUTIONS.*

(FELDTMANN AND BETTEL.)

From 5 to 10 or more grams of the commercial cyanide are dissolved in water, and the insoluble matter, if any, filtered off.

The solution is agitated with a small quantity of precipitated lead carbonate—of course slightly in excess, and filtered. The precipitate, consisting of lead carbonate and sulphide, is transferred to a flask, and covered with a few c.c.'s of a solution of potassic or sodic cyanide free from sulphides, sulphocyanides, or ferrocyanides. This solution may be prepared from pure potassic or sodic hydrate and a solution of pure distilled hydrocyanic acid.

To the mixture in the beaker add hydrogen peroxide in slight excess—*i.e.*, three or four times as much as is needed to whiten the precipitate. (The hydrogen peroxide for this purpose should be purified by agitation with ether and evaporation of the ether in a water-bath.)

A small quantity—say $\frac{1}{2}$ gram—of manganese peroxide is then added, and the mixture agitated for about two minutes, after which the solution is filtered off, acidified with sulphuric acid, and titrated with $\frac{N}{100}$ potassic permanganate.

1 c.c. of $\frac{N}{100}$ potassic permanganate equals 0·000053 grm. sulphur or 0·000182 grm. potassic sulphide.

The potassic permanganate may be standardized by means of pure potassic sulphocyanide.

1 c.c. = ·0001618 grm. KCyS.

* Paper read before the Chemical and Met. Society of S.A.

Estimation of Hydrocyanic Acid.—To 50 c.c. of the solution add a solution of bicarbonate of potash or soda, free from carbonate, or excess of carbonic acid. Titrate as for KCy. Deduct from this the amount of KCy found. The difference equals the amount of HCy present.

$$1 \text{ c.c. AgNO}_3 = 0.0414 \text{ HCy.}$$

Estimation of double Cyanides.—Add excess of pure caustic soda to 50 c.c. of solution, and a few drops of KI solution, then titrate with AgNO_3 . Deduct KCy and HCy, as found above; the difference is $\text{K}_2\text{Zn}(\text{Cy})_4$ as KCy, less 7.9 per cent.

The KCy as found here is calculated to $\text{K}_2\text{Zn}(\text{Cy})_4$, as under:—

$\text{KCy} \times 0.9493 = \text{K}_2\text{Zn}(\text{Cy})_4$. Add to this 7.9 per cent. of total, or for every 92.1 parts add 7.9 parts.

Estimation of Ferro- and Sulpho-Cyanides.—When organic matter is present, shake with powdered quicklime and filter.

A burette is filled with the cyanide solution for analysis, and run into 10 or 20 c.c. $\frac{1}{100}$ Normal $\text{K}_2\text{Mn}_2\text{O}_8$ strongly acidified with H_2SO_4 until the colour is just discharged.

A solution of ferric sulphate or chloride is acidified with H_2SO_4 and 50 c.c. of the cyanide solution poured in. After shaking for about half a minute the Prussian blue is separated from the liquid by filtration, and the precipitate washed. The filtrate is next titrated with $\frac{1}{100}$ normal $\text{K}_2\text{Mn}_2\text{O}_8$.

Let x be c.c. permanganate required to oxidize ferro-cyanide, then $x = y - z$.

$$(x) \ 1 \text{ c.c. } \frac{1}{100} \text{ normal } \text{K}_2\text{Mn}_2\text{O}_8 = 0.003684 \text{ grm. } \text{K}_4\text{Fe}(\text{Cy})_6.$$

$$(z) \ 1 \text{ c.c. } \frac{1}{100} \text{ normal } \text{K}_2\text{Mn}_2\text{O}_8 = 0.0001618 \text{ grm. } \text{KCyS.}$$

THE ANALYSIS OF CYANIDE SOLUTIONS.*

(BY LEONARD M. GREEN, A.R.S.M.)

The tests used depend essentially on alkalimetric determinations. In Mr. Green's scheme the following constituents are estimated:—

- | | |
|--|------|
| (1.) The total cyanide | = T. |
| (2.) The protective alkali | = p. |
| (3.) The alkaline or alkaline-earth hydrates | = h. |
| (4.) Alkaline mono-carbonates | = N. |
| (5.) The ferrocyanides | = S. |
| (6.) The zinc | = Z. |

* Extracts from Paper read before the Inst. Min. and Met., London.

The methods depend on the facts that :—

- (1.) Potassium ferrocyanide is neutral to phenolphthalein.
- (2.) That 1 c.c. of decinormal potassium ferrocyanide precipitates 0.75 c.c. decinormal zinc from a neutral dilute solution of a zinc salt.
- (3.) That when a dilute neutral solution of a zinc salt is precipitated by adding an excess of sodium carbonate solution, the excess of alkali being afterwards carefully neutralized to phenolphthalein, by the addition of decinormal acid, a precipitate of basic zinc carbonate of almost constant composition is obtained. The precipitate obtained in this way is the normal basic carbonate.
- (4.) That zinc hydrate or carbonate, when treated with an excess of potassium ferrocyanide, forms zinc ferrocyanide and potassium hydrate or carbonate, the alkalinity produced being proportional to the precipitate acted on. This reaction does not immediately proceed to the end, as at first only a portion of the alkalinity is formed; but if this be neutralized with acid a further amount of alkali is formed, and so on to the finish, the reaction taking a little time for completion.

The Actual Analysis.—(1.) The first estimation is that of the total cyanide. This is performed in the usual way by adding to the solution to be tested an excess of caustic soda and a little potassium iodide, and titrating with silver nitrate till a distinct, permanent yellowish cloudiness appears. It must be noted that the end-point is not reached until there is a *distinct* yellowish cloudiness.

Where much zinc and ferrocyanides are present, a faint, white cloudiness, probably due to the precipitation of zinc-ferrocyanide, is sometimes produced before the true "end-point." This must be disregarded, the true "end-point" occurring when the yellowish cloudiness, due to silver iodide, is permanently formed. This is the only definite end-point in titrating such a solution, and a large excess of sodium hydrate does not appreciably alter it.

(2.) In the second test the alkaline and alkaline-earth hydrates plus half the mono-carbonates, viz., the "protective alkali," is determined. This test is a simple alteration of Clennel's.

Excess of potassium ferrocyanide is added to the solution, and then twice the amount of silver nitrate necessary to indicate the total cyanide, viz., sufficient to precipitate the whole of the cyanide. A slight excess does not matter, as it merely precipitates some chloride, or, if no chlorides are present, some sulphocyanide or ferrocyanide. The zinc all occurs in the precipitate as

ferrocyanide, and the alkaline hydrates and carbonates are left in solution.

Phenolphthalein is then added, and the solution titrated with decinormal nitric acid till colourless, or till it acquires the faint greenish yellow tinge produced by the excess of ferrocyanide. The result indicates the "protective alkali."

Usually an identical result is obtained by only adding a little more than the amount of silver nitrate necessary to indicate the total cyanide, but this leaves potassium silver-cyanide in solution, and towards the close of the acid titration there is a slight chance of its being acted on by the acid forming AgCy , and setting free HCy , which would slightly increase the acid required, and obscure the end-point.

(3.) In the third test the alkaline hydrates are estimated. An excess of barium chloride is first added to the solution (sufficient to precipitate the sulphates and carbonates), and then the procedure in the last test is repeated. The result obtained indicates only the alkaline hydrate.

In the event of no hydrates being present, bicarbonates probably exist in solution. They may be estimated by adding a known amount of standard sodium hydrate, and repeating the test, allowing, in calculating the results, for the amount of hydrate added.

(4.) The next estimation is that of the total cyanide + chlorides + sulphocyanides + ferrocyanides + any other salt precipitated by silver nitrate before the precipitation of chromate. This test, though not important in itself, is necessary for the subsequent determination of the zinc.

The amount of acid used in the second test is first added to the solution (viz., sufficient to neutralize the protective alkali). Then one drop of a strong solution of potassium chromate is added, and the solution is titrated with silver nitrate till there is a faint permanent reddish colouration.

Towards the end of the titration, the reddish colouration appears, and only goes again slowly on shaking and standing a few seconds. This is due to the fact that the action of the silver nitrate on the precipitated zinc ferrocyanide is slow, and consequently some silver chromate is temporarily formed; this chromate, however, when well shaken up with the zinc ferrocyanide, is decomposed, and silver ferrocyanide is formed. The end-point therefore only occurs when the reddish colouration is decided and permanent.

(5.) The fifth estimation is that of the zinc - the ferrocyanide. Sufficient sodium carbonate is added to supply enough carbonic acid for the basic zinc precipitate in case there should not be enough already in the solution. Then twice the amount of silver nitrate necessary to indicate the total cyanide is added and the

whole is well shaken. All the cyanide is now precipitated as silver cyanide, and any slight excess of silver nitrate will merely precipitate some chloride.

The zinc will have been partially precipitated as ferrocyanide by the ferrocyanide originally present in solution, but the remainder will have been precipitated as a basic carbonate, and there will be an excess of alkali in the solution. This alkalinity is neutralized by the addition of decinormal acid. During this neutralization the basic zinc carbonate gradually acquires its normal composition $3\text{Zn}(\text{HO})_2 \cdot 2\text{ZnCO}_3$, and the neutralization must not be hurried.

The colour of the clear solution above the precipitate must not show a trace of pink, even on standing a minute or two, and the contents of the flask should be frequently shaken. In fact, when neutralization is apparently complete, it is best to add another c.c. of acid, shake up well and then add from a burette, a drop or so

at a time, a solution of sodium carbonate (roughly $\frac{N}{10}$ is a convenient strength), till the clear solution again just shows a pink colour.

All the zinc which is not precipitated as ferrocyanide is now in the precipitate as basic carbonate, and the solution is neutral to phenolphthalein. If now an excess of potassium ferrocyanide be added, the basic zinc carbonate reacts with the ferrocyanide, forming zinc ferrocyanide and alkalinity proportional to the amount of basic carbonate present.

This alkalinity is then titrated with decinormal acid, the result representing the zinc less what has been precipitated by the ferrocyanide originally present.

Towards the end of the titration the colour is discharged and returns again slightly, so that a little time is necessary for the determination.

(6.) The last determination is of the zinc. Sodium carbonate is added as in the previous test; then silver nitrate solution is run in, in the same amount as was used in test No. 4, and the flask well shaken. The cyanides, chlorides, sulphocyanides and ferrocyanides will then have been precipitated as silver salts, and the whole of the zinc will have been precipitated as a basic carbonate, an excess of alkali remaining in the solution. This is naturalized precisely as in the last test, an excess of ferrocyanide added, and the alkalinity produced titrated with decinormal acid.

This result consequently represents alkalinity proportional to the whole of the zinc. It includes metals acting similarly to zinc, such as cadmium, and copper if present in small quantity.

Summary.—The following is a short summary of the tests described, with the quantities of solution taken, etc. :—

(1.) Take 50 c.c. of the solution. Add excess of sodium hydrate and a little potassium iodide solution. Titrate with silver nitrate till there is a distinct permanent yellowish cloudiness.

Result = T c.c.

(2.) Take 50 c.c. of the solution. Add excess of potassium ferrocyanide solution. Run in 2T c.c. of silver nitrate. Add phenolphthalein and titrate with $\frac{1N}{10}$ nitric acid till colourless.

Result = p c.c.

(3.) Take 50 c.c. of solution. Add excess of $BaCl_2$ solution, and then excess of potassium ferrocyanide. Run in 2T c.c. of silver nitrate, add phenolphthalein and titrate with $\frac{N}{10}$ nitric acid till colourless.

Result = h c.c.

(4.) Take 50 c.c. of the solution. Add 2T c.c. of silver nitrate, then p. c.c. of $\frac{N}{10}$ nitric acid. Add one drop of a strong solution of potassium chromate, and continue titrating with silver nitrate till there is a faint permanent reddish colouration.

Result (Total $AgNO_3$ added) = Nc.c.

(5.) Take 50 c.c. of solution. Add about 10 c.c. of sodium carbonate solution (roughly decinormal). Run in 2T c.c. (see Test 1) of silver nitrate, and shake well. Add phenolphthalein, and neutralize with $\frac{N}{10}$ nitric acid till the clear solution is colourless.

Shake well at intervals during the neutralization. Add about 1 c.c. more $\frac{N}{10}$ nitric acid, shake up, and then add a solution of sodium carbonate (roughly $\frac{N}{10}$ is a convenient strength), drop by drop, till the clear solution is just faintly pink.

Add excess of potassium ferrocyanide. The solution becomes strongly alkaline to phenolphthalein.

Titrate with $\frac{N}{10}$ nitric acid till colourless.

Result = S c.c.

(6.) Take 50 c.c. of solution. Add about 10 c.c. of sodium carbonate solution. Run in N c.c. of silver nitrate (see Test 3), and shake well.

Add phenolphthalein, and neutralize as in the previous test.

Add excess of potassium ferrocyanide. The solution becomes strongly alkaline to phenolphthalein.

Titrate with $\frac{N}{10}$ nitric acid till colourless.

Result = Z c.c.

NOTES.—If the standard solutions used are $\frac{N}{10}$ nitric acid, and a solution of silver nitrate containing 13.05 gram per litre, then the following are the factors for the different results :—

- (1.) $T \times 0.02\% = \text{Total Cyanides estimated as KCy.}$
- (2.) $p \times 0.0112\% = \text{Protective Alkali estimated as KHO.}$
- (3.) $h \times 0.0112\% = \text{Alkaline hydrates estimated as KHO.}$
- (4.) $p - h \times 0.0276\% = \text{Alkali carbonates.}$
- (5.) $Z - S \times 0.0351\% = \text{Potassium ferrocyanide.}$
- (6.) $Z \times 0.0081\% = \text{Zinc.}$

The presence of sulpho-cyanides or ferrocyanides in small quantity does not interfere with these tests. The latter in large quantity does, as precipitates of zinc and silver ferrocyanides rapidly destroy the indicator phenolphthalein although potassium ferrocyanide itself does not appear to do so.

A large excess of sodic carbonate has hardly any effect on the final result of the titration for zinc, but it tends to decrease the sharpness of the end points, and increases the time taken for the reactions.

THE ESTIMATION OF OXYGEN IN WORKING CYANIDE SOLUTIONS.*

(BY ANDREW F. CROSSE.)

Prepare a solution containing :—

20 grams Iodide of Potassium,
2 „, Nitrate of Potassium,

dissolved in distilled water and made up to 100 c.c.

The sample of cyanide solution can be taken in an ordinary Winchester quart bottle filled perfectly full.

Syphon off a sample into a separating funnel containing from 250 to 300 c.c., then add 1 c.c. of the solution of iodide and

* *Jour. Chem. and Met. Soc., S.A.*, May 1902.

nitrate of potassium, and 3 c.c. of dilute sulphuric acid (1-1). After shaking up and allowing to stand for 15 minutes, titrate in an atmosphere of coal gas with a hyposulphite of sodium solution containing 7.75 grams per litre, 1 c.c. of which is equal to 0.25 m'grams of oxygen for the iodine liberated.

The peculiarity of the reaction is that no iodine is apparently liberated, and the cyanide solution remains colourless. But iodine has been liberated all the same to form a very unstable compound, iodide of cyanogen, and the amount is in exact proportion to the free oxygen plus the quantity to be subsequently allowed for, and can be estimated by the hyposulphite solution, using a starch solution as an indicator. Of course it is necessary to estimate the correction required for the iodine liberated by the reagents, and also in case any nitrates were present in the cyanide solution.

This correction is easily made in the following manner:—

Take about 400 c.c. of the solution under examination, add to it 0.3 grams pure ferrous sulphate and the same weight of caustic lime, shake up well and filter into a flask through which coal gas is passing, the precipitated ferrous hydrate will have absorbed all the free oxygen in the cyanide solution.

A good many experiments were made to test the above method and the following will be a good example:—

A Winchester quart bottle was partly filled with a sample of an ordinary working solution, the bottle was well shaken for some time in order to saturate the solution with oxygen.

The determination was made as described, using a pipette containing 292 c.c. of solution, 13.4 c.c. of hyposulphite of soda was required to unite with the iodine liberated.

Then 400 c.c. of the cyanide solution was freed from oxygen and the amount of hyposulphite required was 6.0 c.c.

So that 13.4—6.0 leaves 7.4 c.c. of hyposulphite required for the iodine liberated by the free oxygen in 288 c.c. of solution or 1.85 m'grams oxygen in 288 c.c., or 6.4 m'grams oxygen per litre.

In connection with the oxygen determination Mr. Crosse has made some interesting experiments on the amount of oxygen absorbed by sands and spitzluten concentrates.

He says: "I take, say, from 200 to 500 grams of the material under examination and shake it up for some hours in a large bottle full of a cyanide solution of known quantity and saturated with oxygen and then determine the amount absorbed or lost per litre, and so calculate out the amount of oxygen required by each kilogramme or ton of sands. I would, however, remark that though oxygen is being absorbed by certain substances during treatment, such as ferrous sulphides, etc., gold is also being dissolved, but more slowly and to a lesser degree than if no oxygen

absorbing matters were present. If we have ferrous hydrate present, which would almost immediately take up the oxygen, then no gold would be dissolved, but certain substances act slowly, allowing some gold to be dissolved at the same time that oxygen is being absorbed. If this were not the case it would often happen that no gold would be dissolved till several solutions had passed through the sands."

DETERMINATION OF ZINC AND BASE METALS IN A CYANIDE SOLUTION.*

(By ANDREW F. CROSSE.)

The following is an accurate method for determining copper, iron, zinc and nickle in solutions:—

Take from 500 to 1000 c.c. of solution, acidify with a slight excess of sulphuric acid, add five or six grams of pure acid sulphate of potassium, and evaporate to dryness in a platinum dish and then heat to dull redness in order to melt the mass. The metals are obtained as sulphates and can be separated and estimated in the usual way.

ESTIMATION OF ZINC.

In the solutions on the Rand, where we are not usually troubled with anything more than a trace of copper, the chief metallic ingredient is necessarily zinc and the estimation of this metal would often be useful.

Take 300 c.c. of solution, add about a gram of cyanide of potassium and the same quantity of pure caustic potash or soda, heat nearly to boiling point and then add a slight excess of sulphide of sodium in solution. The zinc will be quickly precipitated as a sulphide, and should be collected on a filter paper and washed with hot water. Then place the filter paper in a wide mouthed bottle of known capacity between 250 and 300 c.c.

This bottle must be provided with a well fitting india-rubber bung through which a moderately wide tube is inserted about 8 to 10 inches long.

Then fill up the bottle with a weak solution of pure ferric sulphate containing 5 to 7 per cent. of sulphuric acid, place the bottle or flask in a bowl of cold water and raise the temperature to boiling point. The reason for the glass tube will be apparent as it allows for the expansion of the liquid.

* *Jour. Chem. and Met. Soc., S.A., May 1902.*

The zinc sulphide will have decomposed and reduced a proportionate amount of ferric sulphate to ferrous sulphate. When nearly cold, filter off the solution through a dry filter paper, and take half the quantity contained in the bottle and titrate with decinormal permanganate of potassium.

1 c.c. = .00325 grams of zinc. Sulphide of zinc is, however, slightly soluble in weak cyanide solutions, but having made various experiments, it was found that one milligram of zinc per 100 c.c. of solution taken, would be the right amount to add as a correction to the results obtained.

Various solutions were prepared containing known quantities of the double cyanide of zinc and potassium with ferro-cyanide and sulpho-cyanide of potassium, and the results obtained were very satisfactory.

NOTES ON THE ESTIMATION OF PROTECTIVE ALKALI IN CYANIDE SOLUTIONS.*

(BY ANDREW F. CROSSE.)

Pure cyanide of potassium becomes rapidly decomposed if brought into contact with air, owing to the presence of carbonic acid. Most South African tailings and slimes contain a certain small percentage of free sulphuric acid, or basic ferric salts, which decompose cyanide of potassium. We must protect or prevent this decomposition as far as possible. Cyanide of potassium is an expensive material; we should therefore endeavour to prevent its unnecessary decomposition, by the use of some inexpensive alkali—such as lime—and it becomes necessary to have an accurate method for determining the amount of caustic lime or other protective alkali in solution.

Protective alkali is the alkali present which will unite with any acid before decomposition of the cyanide begins.

It is very important in many cases to be able to determine the percentage of this alkali exactly.

In my experiments lime (calcium oxide) is taken as the protective alkali, but the results can be calculated out for caustic potash or caustic soda, if so desired.

If an exact amount of sulphuric acid be added to a solution of pure cyanide of potassium containing the latter in excess, we have the following reaction:—

$2\text{KCN} + \text{H}_2\text{SO}_4 = \text{K}_2\text{SO}_4 + 2\text{HCN}$ or 98 parts H_2SO_4 liberate 54 parts of hydrocyanic acid.

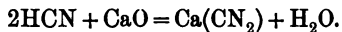
Now, if instead of H_2SO_4 we take pure bi-sulphate of potassium, the following reaction takes place:—

* *Trans. Chem. and Met. Soc., S.A.*, 1899.

$\text{KHSO}_4 + \text{KCN} = \text{K}_2\text{SO}_4 + \text{HCN}$ or 136 parts KHSO_4 liberate 27 parts HCN or one gram KHSO_4 will liberate 1.985 grams HCN from a solution containing an excess of cyanide of potassium.

Supposing, however, that the cyanide of potassium solution contains a free alkali, the sulphuric acid is neutralized, and less hydrocyanic acid is liberated, in direct proportion to such alkali present. If there be no protective alkali, I know that in presence of an excess of KCN one gram of KHSO_4 liberates 1.985 grams HCN.

If less HCN is liberated it practically amounts to its being retained by the alkali, so that if I take the equation—



54 parts of HCN = 56 parts Calcium Oxide.

The proposition is now as follows :—

Take 500 c.c. of a cyanide solution, add one gram of KHSO_4 , boil for 45 minutes, and collect the HCN in a flask containing caustic potash, passing the vapour through a Liebig's Condenser.

I estimate the HCN in the ordinary way in the potash solution with nitrate of silver.

Let A = amount of HCN liberated by one gram KHSO_4 in presence of excess of cyanide without any alkali.

Let B = amount found in experiment with alkali.

$A - B = C$ $C \times \frac{4}{5}$ or $1.037 =$ amount of calcium oxide present.

I made a series of experiments with various proportions of carefully prepared lime water and cyanide solutions, and obtained very accurate results—as for instance, the following. I took 300 c.c. distilled water containing two grams of pure cyanide of potassium, and added 200 c.c. of saturated lime water (at 18° Cent.) which contains 0.260 grams calcium oxide. On treating the solution as described I obtained the following results :—

I had taken in this case 2 grams KHSO_4 .

HCN liberated by 2 grams $\text{KHSO}_4 = 0.3970$ grams.

HCN liberated in experiment = 0.1509 „

0.2461 grams.

$C = 0.2461 \times 1.037 = 0.2552$ grams.

Calcium oxide found 0.255 „

„ „ taken 0.260 „

or in percentages 0.051 and 0.052 respectively or only $\frac{1}{1000}$ of a per cent. from the correct amount, or within 2 per cent. of the lime taken. Of course the operation requires care and practice.

GRAM TABLE.

FOR THE ASSAY OF CYANIDE SOLUTIONS.

If $\frac{1}{2}$ -pint of Solution gives Fine Metal:	One Ton of Solution will give Fine Metal.			If $\frac{1}{2}$ -pint of Solution gives Fine Metal:	One Ton of Solution will give Fine Metal.		
Gram.	ozs.	dwt.	grs.	Gram.	ozs.	dwt.	grs.
·0001	0	0	5·5	·0200	2	5	20
·0002	0	0	11	·0300	3	8	18
·0003	0	0	16·5	·0400	4	11	16
·0004	0	0	22	·0500	5	14	14
·0005	0	1	3·5	·0600	6	17	12
·0006	0	1	9	·0700	8	0	10
·0007	0	1	14·5	·0800	9	3	8
·0008	0	1	20	·0900	10	6	6
·0009	0	2	1·5	·1000	11	9	4
·0010	0	2	7	·2000	22	18	8
·0020	0	4	14	·3000	34	7	12
·0030	0	6	21	·4000	45	16	16
·0040	0	9	4	·5000	57	5	20
·0050	0	11	11	·6000	68	15	0
·0060	0	13	18	·7000	80	4	4
·0070	0	16	1	·8000	91	13	8
·0080	0	18	8	·9000	103	2	12
·0090	1	0	15	1·0000	114	11	16
·0100	1	2	22	2·0000	229	3	8

GRAIN TABLE.

FOR THE ASSAY OF CYANIDE SOLUTIONS.

Grains.	ozs.	dwt.	grs.	Grains.	ozs.	dwt.	grs.
·001	0	0	3·5	·060	0	8	23
·002	0	0	7	·070	0	10	11
·003	0	0	11	·080	0	11	23
·004	0	0	14·5	·090	0	13	10
·005	0	0	18	·100	0	14	22
·006	0	0	21·5	·200	1	9	20
·007	0	1	1	·300	2	4	18
·008	0	1	4·5	·400	2	19	16
·009	0	1	8	·500	3	14	14
·010	0	1	12	·600	4	9	12
·020	0	3	0	·700	5	4	10
·030	0	4	12	·800	5	19	8
·040	0	6	0	·900	6	14	6
·050	0	7	11	1·000	7	9	4

TABLE FOR 8 OZS. = $\frac{1}{2}$ LB. OF SOLUTION.

If $\frac{1}{2}$ lb. of Solution gives of fine metal grains.	One ton will give of fine metal			If $\frac{1}{2}$ lb. of Solution gives of fine metal grains.	One ton will give of fine metal		
	ozs.	dwt.	grs.		ozs.	dwt.	grs.
·001	0	0	4·5	·080	0	14	22
·002	0	0	8·75	·090	0	16	15
·003	0	0	13·25	·100	0	18	8
·004	0	0	18	·200	1	16	16
·005	0	0	22·5	·300	2	15	0
·006	0	1	2·5	·400	3	13	8
·007	0	1	7·25	·500	4	11	16
·008	0	1	12	·600	5	10	0
·009	0	1	16	·700	6	8	8
·010	0	1	20	·800	7	6	16
·020	0	3	16	·900	8	1	0
·030	0	5	12	1·000	9	3	8
·040	0	7	11	2·000	18	6	16
·050	0	9	4	3·000	27	10	0
·060	0	11	0	4·000	36	13	8
·070	0	12	23	5·000	45	16	16

SOME USEFUL CONSTANTS.

The cubic content of a circular vat in feet = Dia^2 in feet
 $\times .7854 \times \text{depth in feet.}$

1 cubic foot of water = $62\frac{1}{4}$ lbs. nearly.

1 ton of water contains about 36 cubic feet.

1 gallon equals 10 lbs.

1 lb. avd. equals 7000 grains.

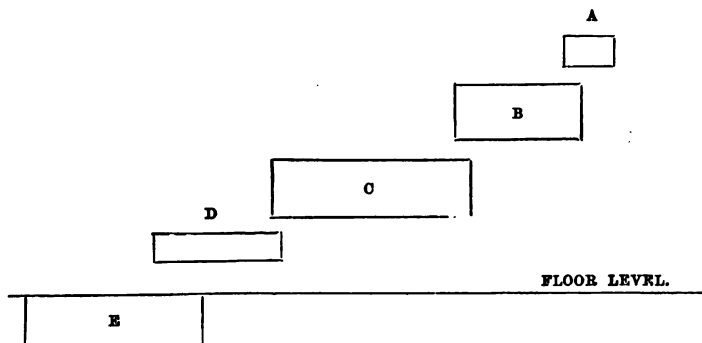
To convert lbs. troy into lbs. avoirdupois $\times .82286$.

To convert lbs. avoirdupois into lbs. troy $\times 1.2153$.

CHAPTER V.

THE APPLIANCES AND PLANT FOR CYANIDE EXTRACTION.

THE appliances in use are of much the same nature at all cyanide plants, but their size, shape, and arrangement are subject to endless variations, being chiefly affected by local conditions, the character of the material to be treated, and the individual taste or fancy of the metallurgist. In all cases the designer should utilize the natural advantages at his disposal ; and, where possible, the solution vat, leaching vats, vacuum-cylinder, storage tanks, zinc extractors, and sumps should be placed on three separate tiers or platforms, so as to permit of the circulation of the solutions by gravitation, as shown in the following diagram :—



A—DISSOLVING TANK. B—SOLUTION TANK.
C—LEACHING VAT. D—EXTRACTOR. E—SUMP.

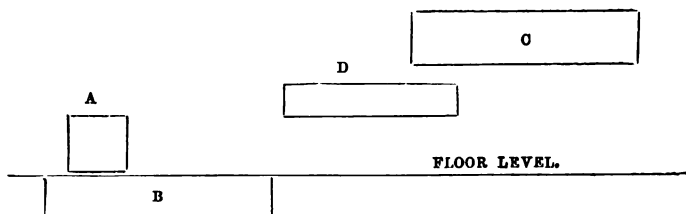
When a vacuum-cylinder is used, a storage tank must be placed below it to receive the solution when the cylinder becomes full. The bottom of the storage tank must be above the extractor.

Whatever arrangement is adopted, the solution has to be pumped from the sumps, either to the solution tank B, or directly

to the leaching vat. In the latter case the cyanide solution is made up to the working strength in the sump, the dissolving tank A being placed on the floor level; or the strength of the sump solution having been ascertained, the required volume is pumped up to the leaching tank and brought up to working strength by dissolving the requisite quantity of cyanide, placed, in a perforated box or tray, at the discharge end of the solution pipe.

Where the height necessary for gravitation cannot be obtained, the need of constructing high platforms, with a correspondingly high building, is obviated by making up the working solution in one of the sumps, with each of which the dissolving tank and solution sump is connected. To do this without inconvenience an extra sump should be provided.

This arrangement is shown in the following diagram :—



A—DISSOLVING TANK. B—SUMP, ALSO USED AS SOLUTION TANK.
C—LEACHING VAT. D—EXTRACTOR.

The necessary appliances for a successful and well-equipped cyanide plant, where the ore is to be treated by percolation, are as follows :—

1. A dissolving tank.
2. Solution vats.
3. Leaching or percolating vats.
4. Vacuum-cylinder and air-pumps if percolation slows down.
5. Storage vats.
6. Zinc precipitation boxes.
7. Sumps.
8. Solution pump.
9. Fresh-water pipes connected with every tank and vat.
10. Line of pipes connecting vacuum-pump and cylinder with leaching vats.
11. Line of pipes connecting solution pump with each sump, leaching vat, and solution tank.

12. Lines of weak and strong solution pipes connecting leaching vats with extractors.
13. Filter vat for washing and drying gold precipitates.
14. Melting and roasting furnace.
15. Assay office and laboratory.

In the case of double treatment, or intermediate filling, additional vats are required. They are generally placed over, or at a higher elevation than the leaching vats.

Dissolving Tank.—This is constructed of wood, iron, or steel. When made of wood, the staves are of 2 in. or $2\frac{1}{2}$ in. pine; and when of iron or steel, the plates are from $\frac{1}{8}$ in. to $\frac{1}{4}$ in. thick, according to the size of the vat, and stiffened with angle-iron and hoops. The size varies from 3 ft. to 6 ft. in diameter, and from $2\frac{1}{2}$ ft. to 4 ft. in depth.

In large cyanide works a perforated tray, to hold the solid cyanide salt, is suspended over the tank by a chain or steel-wire rope, running over a pulley fixed to a beam overhead. The end of the rope or chain passes over a second pulley on the same beam, fixed at a sufficient distance to permit a balance-weight on the end of the rope to clear the side of the tank.

In practice the solid salt is taken out of the original packing case and cleaned, by removing all adhering particles of sawdust or other packing material with a husk-broom. It is then broken into small pieces and placed in the perforated tray, which is allowed to subside into the solution. The rapid dissolution of the cyanide salt can be effected by imparting motion to the tray by pulling at the weighted end of the rope.

The discharge-hole from the dissolving tank should always be placed three or four inches above the bottom, so as to allow a settling space for the impurities contained in the cyanide.

The impurities contained in commercial cyanide consist principally of black carbide of iron, and other insoluble matters, which would, if permitted, obstruct the solution pipes, choke up the filter webs, and thus cause vexatious delays. Besides this, carbide of iron decomposes the potassium cyanide solution of gold; hence its presence in the solutions would tend to cause a loss by precipitating a portion of the gold.

Solution Vats.—These are used for making up the cyanide solutions to the working strength. They are open circular tanks from 14 ft. to 20 ft. in diameter, and from 4 ft. to 14 ft. in depth. They are generally constructed of well-seasoned pine. The sides are made of 5 in. or 6 in. planks, $2\frac{1}{2}$ in. or 3 in. thick.

The bottom is constructed of planks 12 in. by 3 in., bolted and dowelled together independently of the sides. The bolts are placed about 3 ft. apart, and are made of $\frac{3}{4}$ in. or 1 in. round

iron. The washer and nut are well countersunk into the plank, and after being tightened, the whole is plugged up with a block of wood to prevent leakage around the bolt. The dowels consist of $\frac{3}{4}$ in. round iron, and are 6 in. or 8 in. long. They are placed about 12 in. apart where the planks come near the circumference.

The bottom is rebated into the side planks, which are kept tight by round iron hoops, $\frac{3}{4}$ in. to $1\frac{1}{2}$ in. in diameter, having three or more cast-iron turn-buckles on each hoop. One, and sometimes two, rings of $\frac{1}{4}$ in. round india-rubber are placed round the bottom before the staves are put on. The rubber rings are placed in grooves which are made with a steel tool.

The side planks, or staves, are kept in their places by the pressure of the hoops alone, which are generally placed from 15 in. to 18 in. apart, with an extra hoop at the bottom. The hoops are placed, on very large deep tanks, only six or seven inches apart at the bottom, where the pressure is greatest, the distance apart gradually increasing to 18 in. or 20 in. at the top. The extra hoop is placed as close to the bottom hoop as the turn-buckles will permit.

One or more solution vats may be required according to the size of the plant.

Leaching or Percolating Vats.—These are made of many different shapes, sizes, and kinds of material. At first, small square tanks of wood were used, but the difficulty of keeping these tight led to the adoption of circular vats, which are also stronger.

In Australia, the favourite material is wood, but in South Africa and New Zealand, steel vats are at present preferred to wooden ones, in all the recently erected cyanide plants. In America, both steel and iron vats are preferred to wooden ones.

In Victoria and New South Wales, leaching vats made of ordinary corrugated iron with wooden bottoms have been used with very satisfactory results. Mr. W. Eddowes, who used them in Victoria, informed the author that they were light, strong, cheap and durable. For vats of large capacity, corrugated iron of No. 16 gauge is employed. The cost is said to be half that of steel vats of the same capacity. When the value of corrugated iron becomes better known, it will doubtless be more largely employed, more especially in outlying mining camps where transport costs are heavy.

The construction of the circular wooden leaching vats is in every respect the same as that of the solution vats already described, differing only in being provided with discharge-doors. The first vats in use at the cyanide works of the Waihi Gold and Silver Mining Company were $22\frac{1}{2}$ ft. in diameter, and 4 ft.

deep. The sides were built of 5 in. kauri planks, 3 in. thick, bound together with five hoops of round iron, three of which were $\frac{7}{8}$ in. in diameter, and two 1 in., having three turn-buckles on each hoop. Five inches is taken off the depth by the false bottom, filter-frame, and cloth. At the Company's new mill, there are ten concrete tanks in two rows, each tank being 50 ft. by 40 ft. and 4 ft. deep.

The foundations or supports of wooden vats, 20 ft. in diameter, with 7 ft. staves, to hold a charge of 70 tons of tailings, designed by the author for the Moanataiari Gold Mining Company, Thames, consisted of *mudsills*, 15 in. by 6 in., laid flat, on which rest the *sole* pieces, 10 in. by 8 in., supporting the *props* or uprights, 8 in. by 8 in., on which are laid the *bearers* of the same dimensions as the sole pieces. There are five rows of props—two rows of three, three rows of five—twenty-one in number altogether, spaced with 4 ft. centres both ways. The bottom of the vat rests on the bearers, which are cut so as to allow a space 4 in. wide between their ends and the side planks. This space is left to detect and repair any leaks that may take place round the sides of the vat.

Where the solid rock is exposed to build upon, the mudsills can be dispensed with. (For detailed drawings, see Plate I.)

The same support can be used for iron or steel vats, but it is advisable to place 9 in. by 3 in. planks across the bearers, so as to more evenly distribute the weight of the vat, the bottom of which rests on the planks.

At the Simmer and Jack Cyanide Works, Johannesburg, the leaching tanks are constructed of pine. They are 42 ft. in diameter, and 14 ft. deep; bound together with fifteen hoops of round iron. They rest on piers of solid masonry. The staves and bottoms are made of 9 in. by 3 in. material.

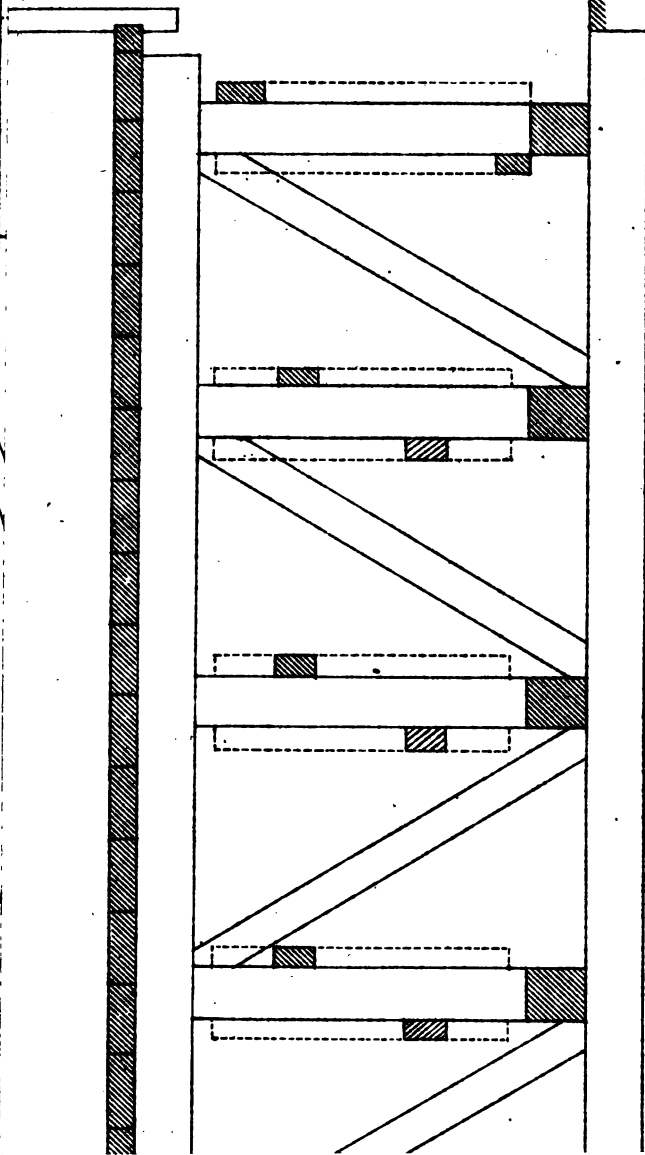
At the "Main Reef" Works there are six leaching vats, each 26 ft. inside diameter, with 8 ft. staves, and holding 135 tons of tailings.

The staves are $4\frac{1}{2}$ in. wide and 3 in. thick, and planed to the level by machines, and afterwards hand-dressed on the abutting edges. They are checked $\frac{3}{4}$ in. to fit on the bottom, with a chime 6 in. below the check.

The bottom of the vat is made of 9 in. by 3 in. deals, planed by machine and grooved $\frac{3}{8}$ in. by $\frac{1}{2}$ in. by a saw, and is also hand-dressed on the edges. Clear-pine tongues, 1 in. by $\frac{3}{8}$ in., fill the grooves. The joists across the tunnel, below the vat, consist of 9 in. by 3 in. deals, bolted together in pairs, and laid 2 ft. 3 in. apart from centre to centre.

These joists are first laid in position, then the bottom of the vat

PLATE I





is laid down, cramped up, and the circle struck out. The bottom is now sawn to the circle, and when bevelled all round is ready for the staves, which are driven up as tightly as possible.

Six hoops of round iron are used to keep the staves in their places; the top pair 1 in. in diameter, the middle pair $1\frac{1}{8}$ in., and the lowest pair $1\frac{3}{8}$ in. in diameter, with screwed ends.

Each hoop is made in three sections, rolled to the required curve, and connected by cast-iron turn-buckles.

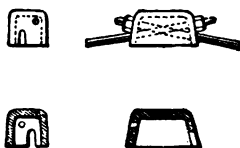


FIG. 1.—SHOWING CONSTRUCTION OF TURN-BUCKLE.

Scale: $\frac{1}{2}$ in. = 1 ft.

The screwed ends pass through the turn-buckles, and while each hoop is being drawn up it is hammered with a heavy sledge-hammer.

Two carpenters, practised at the work, can dress the material for a vat, 28 ft. in diameter and 8 ft. deep, in about a week, and erect it in about four days.

With large vats, constructed of brick and cement, in an excavation in the ground, there is no means of ascertaining what leakage is going on; and, in a process in which gold solutions are being dealt with, an exceedingly small leak, in the course of the year, would represent a considerable loss. For this reason their construction cannot be recommended where material is procurable for the construction of wooden or steel vats.

At the Langlaagte Estate Company's Cyanide Works, S.A., the tanks are round and constructed of brick, faced with hydraulic cement. Their size is 40 ft. in diameter and 10 ft. deep. At the Crown Reef Works the tanks are also of brick, 40 ft. square and 10 ft. deep. At the Waihi-Silverton Works, N.Z., the tanks are constructed of $\frac{3}{16}$ in. steel, being 16 ft. in diameter and 4 ft. deep. At the Moanataiari mine, the steel vats are 20 ft. in diameter and 7 ft. deep. At the Cripple Creek Gold Exploration Company's Works, in Colorado, the vats are made of iron, being 20 ft. in diameter.

For the direct treatment of dry-crushed ore, the leaching vats are seldom over 4 ft. deep, on account of the difficulty of percolation with a greater depth of ore; but, with tailings comparatively free from slimes the depth varies from 8 ft. to 14 ft.

The leaching vats, on account of the enormous weight they hold, must be built on strong firm foundations, so as to prevent settling, and the leakage which would be sure to follow. In South Africa they are often built on piers of stone; and in New Zealand and Australia, where timber is plentiful, on massive frames of wood. Whatever the foundations, there should always be free access to the bottom of the tanks, so as to be able to detect and repair leaks.

Each tank is provided with a separate drain-pipe, $1\frac{1}{2}$ in. or 2 in. in diameter, with two stop-cocks near each other, one over the strong solution launder or pipe, the other over the weak solution launder, leading to their respective zinc extractors. When filtration is assisted by a vacuum, a third stop-cock is provided for the air-pipe.

The sizes of the pipes for charging the vats with the solutions are as follows :—

Vats, 20—24 ft. in diameter,	.	.	$2\frac{1}{2}$ in.
„ 24—32 ft.	„	.	3 in.
„ 32—40 ft.	„	.	4 in.

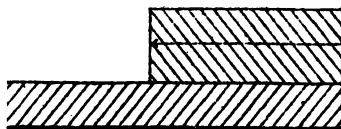
Two or three lines of pipe with two mains, one for the weak and one for the strong; and with three mains, one for the weak, medium and strong respectively) running parallel with the line of leaching vats, afford the simplest, most economical, and effective method of collecting the solutions as they percolate from the vats. This system enables the solutions from each vat to be tested separately and readily, and by this means any mishaps can at once be detected.

Instead of having stop-cocks on the end of the drain-pipe, from each vat, a short length of rubber-hose is sometimes fixed on the end; and by moving the hose the solution can be drained into the strong or weak launder as required.

Steel and iron vats are now coming into general use. They possess many advantages over wooden ones. They are generally coated with a composition consisting of a mixture of coal-tar, pitch, and kauri gum. (See Plate II.)

The filter-frame in steel vats is supported on a ring of iron riveted to the side about 3 in. from the bottom. The filter webbing is laid on the frame and kept in its place by means of a ring of angle-iron, which is constructed in four, six or eight pieces, or lengths, so as to be easily handled. The ordinary method of grouting the cloth between the ring of iron and the side of the vat with a small rope is the best.

In large steel vats, the ring for the reception of the filter-cloth is omitted, and in this case the filter-frame is constructed so as to



Full Size

leave an annular space an inch wide all round the vat so as to permit the filter-cloth to be grouted between the frame and the vat.

Filter-Frames.—The old filter-beds of gravel and sand have been entirely superseded by light wooden frames, over which are placed filter-cloths or webs, consisting of either extra strong Hessian, loose canvas, cocoa-matting, or burlap. For the filtration of slimes, or dry-crushed ores, which always contain a large percentage of very fine sands, a webbing of strong Hessian or canvas is used; and for tailings or concentrates a webbing of cocoa-matting or burlap.

In some plants a duck-cloth filter is laid over cocoa-matting. The duck-cloth for a 100 ton vat costs about £2, 10s. ready made.

At the Waihi Cyanide Works the filter-frames, designed by the general manager, Mr. H. P. Barry, consist of narrow laths placed parallel, and about an inch apart. On these laths are nailed, transversely, narrow moulding-like laths, also about an inch apart. An open frame-work or grating is thus obtained, having openings an inch square.

At the Main Reef Cyanide Works, Johannesburg, the filter frame is made of 3 in. by 1 in. slats, placed on edge, 6 in. apart, their ends being kept 1 in. from the sides of the vat. Strips of wood, 1 in. square, are nailed on the top of the slats 1 in. apart to form a support for the cocoa-matting.

The filter-frames for large vats are constructed in sections. The sections, when fitted together, form a circular frame about an inch less in diameter than that of the vat. This leaves an annular space between the frame and the vat, which permits the filter-cloth to be firmly grouted in its place by means of a small rope passing round the circumference of the vat.

The author has used such filter-frames for over five years, and finds that they possess many advantages over the old gravel-filters.

At plants where intermediate filling is adopted, the intermediate vats are provided with automatic distributors.

Vacuum-Cylinder and Air-Pump.—Filtration is generally assisted by creating an artificial vacuum below the filter-bed. The means mostly adopted in New Zealand and Australia to produce a vacuum is an air-tight boiler, or cylinder, connected with an air-pump. In large cyanide plants two air-pumps and cylinders and storage tanks should be provided.

The cylinders are generally constructed of $\frac{1}{4}$ in. boiler-plate, with $\frac{3}{8}$ in. ends. They are made of different sizes, according to requirements, from 6 ft. to 13 ft. in length, and from 3 ft. 6 in. to 6 ft. in diameter. They are provided with a solution-gauge,

vacuum-gauge, air-cock, and man-hole, as well as the necessary pipe connections.

The air-pump is single or double-acting, 7 in. in diameter, with an 8 in. stroke, making from 80 to 120 strokes per minute, and capable of producing a vacuum of 26 in. of mercury in the cylinder. To prevent heating of the valves it should be surrounded with a water-jacket, through which a current of cold water can continually circulate when the pump is running. The air-pump at the Waihi cyanide plant is 14 in. in diameter, and has a 22 in. stroke.

All the stop-cocks, valves, pipes and connections about the cylinder, air, and solution pumps, and tanks, which are subject to cyanide solutions, should be of black iron.

When the vacuum-cylinder becomes full, the solution is discharged into a storage vat, from which it slowly drains through the zinc extractor. In order to give timely warning when the cylinder became full, the following simple electrical contrivance was used by Mr. Arthur Wilson, the manager, at the Cassel Gold Extracting Company's tailings plant at Waihi.

A small Erdmann float, with a platinum wire fused into the top and coiled into a flat helix, was placed in the solution gauge-tube. Two platinum wires were also fused into the upper end of the gauge-tube, projecting into the tube, opposite to each other, but not in contact. The platinum wires were connected with a small Leclanché battery, and when the float rose in the solution-gauge to the platinum wires, metallic contact was established, and an electric bell in the circuit sounded an alarm.

Discharge of Leached Residues.—Where there is a plentiful supply of water, with a good head, the easiest and cheapest method of discharging the residues from the leaching tanks is to sluice them out by a side-door. At the cyanide plant of the Waihi Gold Mining Company the residues are sluiced out by two 2 in. hose-pipes under a head of 150 ft., giving a pressure of 65 lbs. to the square inch.

At the Witwatersrand Goldfields, where there is a scarcity of water, and often a want of fall for the sludge, the "bottom discharge" is largely practised, the residues being shovelled through a hole in the bottom of the vat into a truck immediately below. At the Barret Company's Works the tailings are shovelled into a launder below the vat, and a stream of water carries them away.

At the Crown, Woodstock and Talisman Cyanide Works in New Zealand, where the ore contains a percentage of coarse gold, the residues are slowly sluiced over extensive amalgamated copper plates placed immediately below the discharge holes.

At the Langlaagte Company's Cyanide Works, near Johannes-

burg, the residues are discharged from the large brick leaching tanks by means of steam travelling cranes, which lower the bodies of the empty trucks into the tanks, where they are filled by Kaffir labour. When filled the trucks are raised and placed on their carriages, to be wheeled away to the dump.

Discharge Doors.—When side-discharge by sluicing is used, one or two outlets are generally provided for each vat; but in the case of bottom-discharge there are two, four, six, or eight discharge openings to each vat, according to its size.

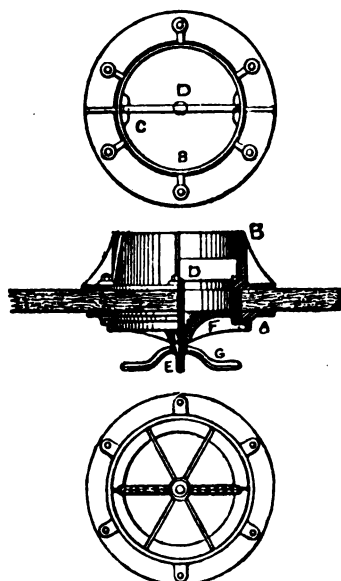


FIG. 2.—BUTTERS' BOTTOM-DISCHARGE DOOR Scale: $\frac{1}{2}$ in. = 1 ft.

At the Witwatersrand Goldfields the bottom-discharge is employed for discharging the round wooden leaching vats. When filling a deep tank with tailings, a length of wrought-iron pipe or cylinder, three or four feet long, is placed over each discharge-hole, and then the tailings are dumped in. The pipe raises the outlet within a few feet of the surface, and thus facilitates the discharge.

On these fields, Butters' bottom-discharge doors are largely used. Fig. 2 shows their construction.

The arrangement is very simple and effective. On the bottom side of the tank, a cast-iron ring, A, is bolted to the cast-iron

cylinder, B, inside the tank. Inside the cylinder is a projecting lug, C, upon which rests the hanger, D, which forms part of the screw, E. The cast-iron cover, F, when placed in position, is fastened by the butterfly-nut, G, and by screwing this firmly the whole arrangement becomes watertight. The faces of the ring and cover should be planed perfectly even, so as to make a good joint. The joint is also made tight by a luting of clay.

Fig. 3 shows another bottom-discharge door, designed by Mr. W. F. Irvine. It is simple in construction and not likely to get out of order, but would be more convenient in shallow than deep vats.

Plate IIA. represents the working drawing of an improved bottom-discharge door, simple in construction, very efficient, and easy to handle. It was originally designed by Mr. G. R. Walker,

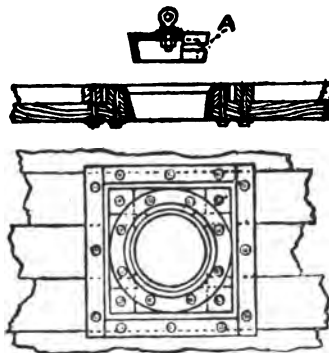


FIG. 3.—IRVINE'S BOTTOM-DISCHARGE DOOR.

A—Recess for Packing. Scale : $\frac{1}{8}$ in. = 4 in.

improved by Mr. A. Price, and perfected and used in its present form by the author in the Moanataiari and other cyanide works. The Koppel patent tank door recently placed in the market differs from the above only in that the pressure to close the door is applied by a screw at the side instead of at the centre.

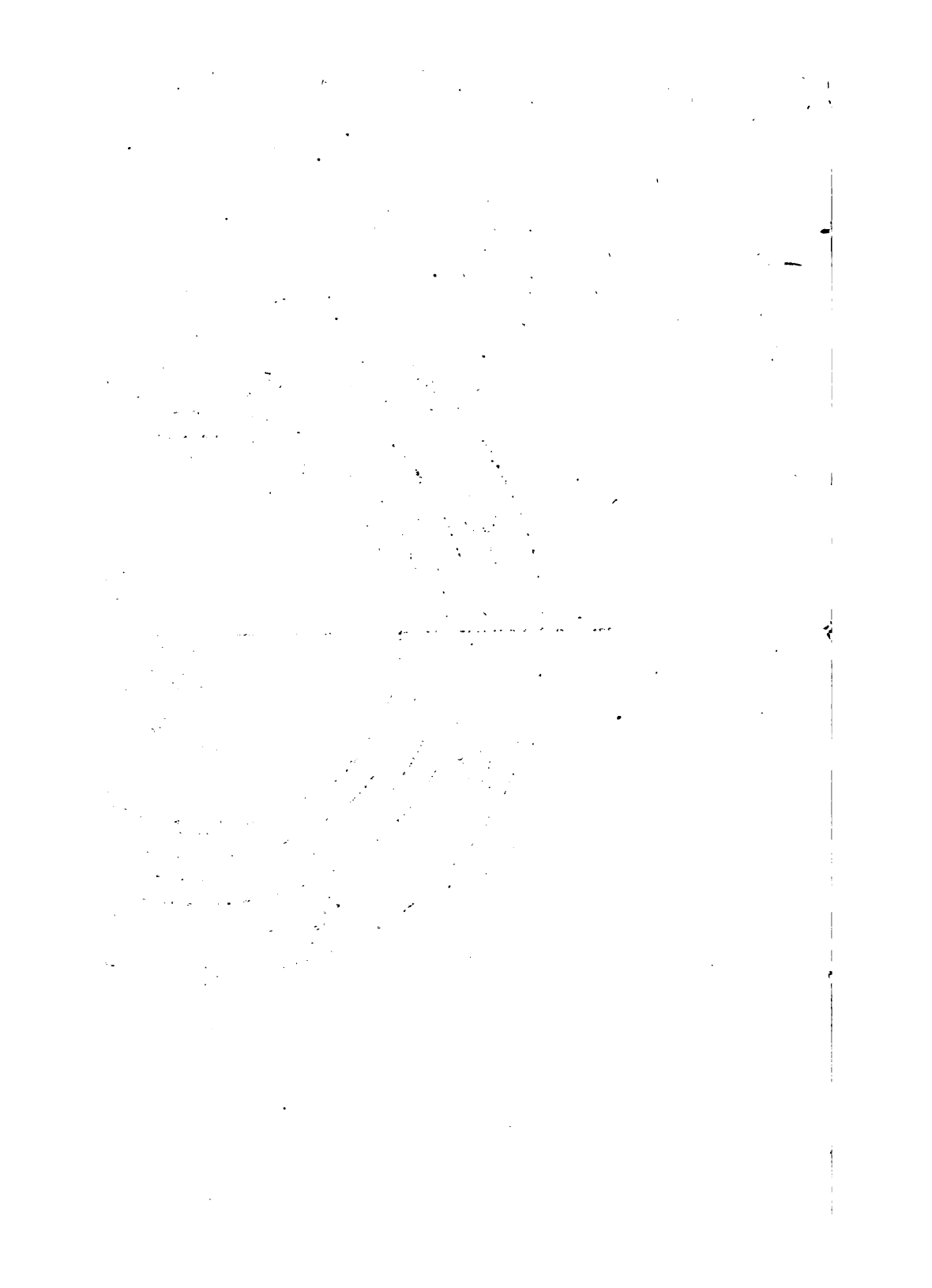
The door is swung on a loose hinge, and faced with a wide ring of rubber-insertion. It is closed by lifting, or swinging the door with the left hand over the discharge hole, and then turning an iron button with the right hand so as to carry the weight of the door, which is now approximately in position. The loose dog is then inserted in the lugs and the door screwed up tightly with both hands. The iron-button turns easily on a small stud-bolt, and serves to keep the door in position preparatory to the final tightening.

-A-

1

1





To open the door the dog is unscrewed, and the button turned to one side, when the door falls open by its own weight, swinging away from the workman. The operation of opening or closing occupies two or three minutes.

The drawing shows a door designed for a steel vat, but it could easily be adapted for a wooden vat by lengthening the cylindrical part, which projects into the vat the necessary length to make it level with the filter-frame, less, of course, the thickness of the coir matting or filter-cloth.

Plate X. in Chapter XI. shows the bottom discharge-door designed by Mr. Roche for the Waihi Company's new cyanide plant at Waikino.

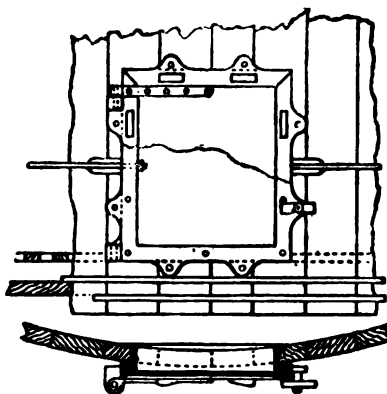


FIG. 4.—SIDE-DISCHARGE DOOR. Scale : $\frac{3}{4}$ in. = 2 ft.

Fig. 4 shows a modification of a side-discharge door for wooden vats, designed by Mr. W. R. Feldtmann, of Johannesburg.

Another method of side-discharge, designed by Mr. Irvine, is in use at the Crown Reef Company's fine cyanide works, where the large square brick and cement vats are provided with doors which permit of the ingress of the discharging trucks.* The door frames are bolted to the cement walls, and the plate-iron doors are drawn tight against these by means of an ingenious arrangement of sliding lugs, bolts, and nuts.

The side-discharge sludge doors used in New Zealand and Australia are of the simplest construction and yet perfectly efficient. They consist of a cast-iron frame with two projecting lugs, one on each side, and a projecting bar running along the top side.

* Feldtmann, *Notes on Gold Extraction*, 1894, p. 3.

The lug on the right side is placed with the notch upwards, that on the left with the notch in the reverse position.

The opening is closed by a cast-iron door, which is kept in position by the pressure of a screw acting through a loose iron dog, the ends of which fit into the lugs so as to obtain the necessary leverage.

The door is suspended in front of the opening, preparatory to fixing up, by a hook of bent round iron, which is supported on the projecting bar on the frame. It is rendered watertight by a facing of rubber insertion, fixed on with tar, or by a luting of clay.

These doors seldom give any trouble. They are easily opened or closed by a few turns of the screw. The different parts are shown to scale on Plate III.

Steel leaching tanks with bottom-discharge, designed by the author for the Moanataiari Cyanide Plant, are shown on Plate II.

Sumps.—There are at least two of these in every cyanide plant, to receive the cyanide solutions after passing through the zinc-extractor, one for the strong solution and one for the weak. In plants dealing with acid ores or tailings there is often an additional tank besides these for storing the alkaline wash-solutions.

The size of the storage sumps depends on the size of the plant. In most plants they are the same size as the leaching vats. They are constructed either of steel, concrete, bricks faced with cement, or wood. The latter is the favourite material. The construction of wooden sumps is the same as that of the storage tanks or vats.

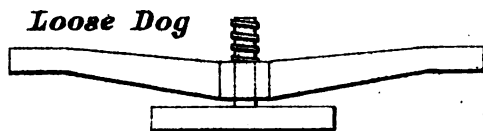
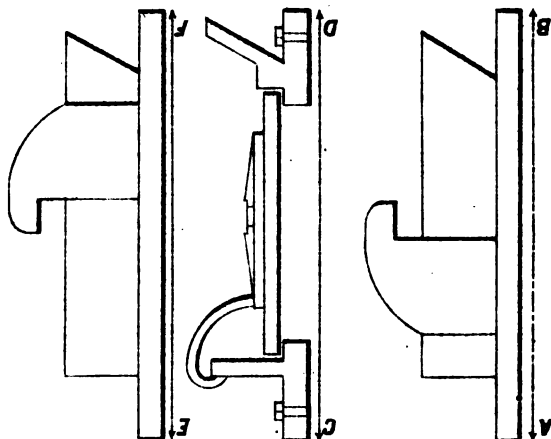
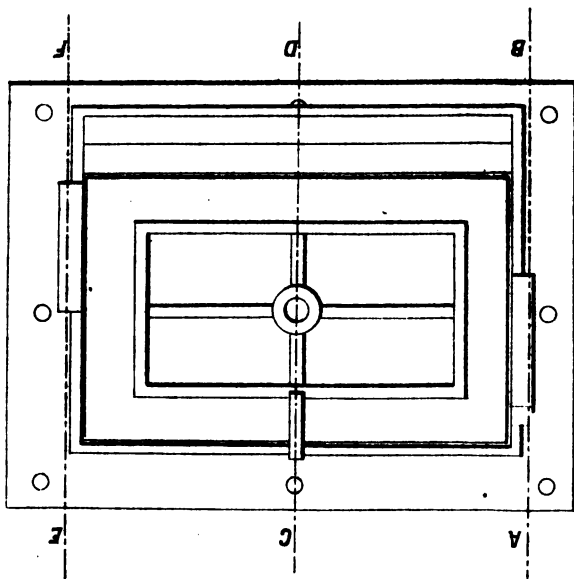
In many plants the sumps are placed below the level of the lower or extractor floor of the cyanide building, and in such cases they are decked over with planks, having a man-hole for repairs or cleaning. The depth of the solution is indicated by the tell-tale.

In some tailings plants the working cyanide solution is made up in strong solution sumps, thus saving the construction of a storage vat, but requiring an extra sump. This method could not be used with advantage for the treatment of dry-crushed ores in which it is necessary to apply the strong solution slowly from below, so as to prevent the formation of lumps and channels in the pulp such as would occur were the solution turned on to the dry ore from above.

Zinc Extractors.—There are at least two of these in every plant, one for the strong solutions and one for the weak. In works treating sands and slimes it is usual to provide a set of three extractors, namely for the strong, medium and weak solutions respectively. They are constructed of wood, and consist of oblong boxes, each divided into a number of compart-

SLUDGE DOOR.

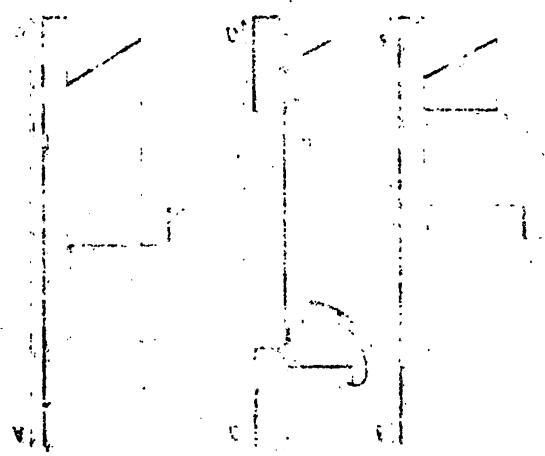
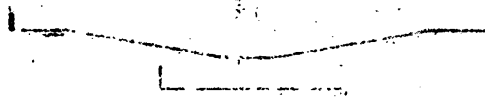
Plate III



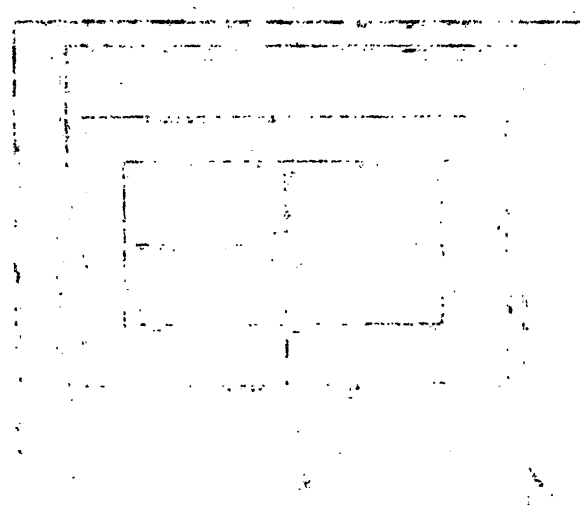
1000

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1000



ments, generally eight, ten, or twelve, by means of partitions and baffle-boards fixed in such a way that the solution finds its way upwards through the zinc-turnings in each compartment.

The first and last compartments are filled with sand or tow to act as filters, the upper one to remove any clayey matter in the solutions, the lower one to prevent the escape of slimes.

The remaining divisions are sometimes provided with removable shallow trays of wood, with convenient handles, often of bent round iron, to enable them to be easily lifted up when necessary. The trays support, at the bottom, a wire screen of $\frac{1}{8}$ in. mesh, which permits the free circulation of the solutions, and enables the gold and zinc slimes, as they form, to fall through into the bottom of the compartment.

Instead of projecting handles, the trays are often provided with an iron plate on two sides with two holes in it to take the ends of hooks for lifting the trays and their contained zinc.

The extractors are cleaned-up from plug-holes on the side, one to each compartment. To facilitate the "clean-up" the bottoms of the compartments slope to the side, and also to the lower side-corner, where the discharge-hole is situated.

A launder of wood or iron is fixed on the discharge-side, immediately under the plug-holes, so as to receive the slimes when they are washed out of the boxes. The top of the extractor, as well as the side launder, is protected with a lid or close-fitting grating of wood or iron, provided with locks, to prevent the contents of the extractor being tampered with.

To facilitate the withdrawal of the wooden plugs from the discharge-holes, short lengths of rubber-hose are fixed in the holes. The rubber is yielding, and, while rendering the holes water-tight, enables the plugs to be withdrawn without the force which is generally necessary when they are driven into the bar-holes.

The extractor should be constructed of well-seasoned pine, with sides of $1\frac{1}{2}$ in. or 2 in. boards, and the divisions of 1 in. boards, well dressed. The size will depend upon the capacity of the plant, and will vary from 12 ft. to 20 ft. in length, from 2 ft. to 3 ft. in depth, and from 15 in. to 45 in. in width. The extractors mostly used are shown on Plates IV. and IVA.

It is now well known that the bulk of the precipitation takes place in the first two or three boxes, and in cyanide plants handling large volumes of cyanide solution daily, the tendency has been of late to reduce the number of divisions in the extractor to six or eight, and to increase their dimensions. In this case the extractor is simply a long box or trough divided by baffle-boards into compartments, without side plugs or side launder. At the periodical "clean-up" each compartment is cleaned separately; or

sometimes two or more, often all the compartments, are made to communicate with each other by plugs at the bottom of baffle-boards so that the slimes can be washed into the lower one of the set thus communicating, and thence be baled out into a tub or run into a bucket. (See Plate IV_A.)

The construction of the extractor permits an endless variety of design according to the individual fancy of the engineer, always bearing in mind that the object to be attained is to compel the solutions to pass upward through the zinc at a uniform and slow rate.

At the Waihi Company's large mill at Waikino, the solutions are collected in a rectangular tank divided into three compartments each forming a coir-filter or clarifying box, for the strong, medium and weak solutions respectively.

Each compartment is provided with a false-bottom, 9 ins. deep, covered with cocoanut matting which is kept in place by a light wooden frame.

From the clarifying boxes or compartments, the solutions run into the different extractor boxes, which are made 23 ft. long by 3 ft. wide by 2 ft. 6 ins. deep. The extractors have flat bottoms and are divided into nine compartments, namely seven for precipitation and two for settling, one at each end, which is filled with tow.

The trays are 6 in. deep and covered with $\frac{1}{8}$ in. mesh screens. The slimes are washed into the end compartment through plugged holes in the baffle boards.

The solutions leave each box by a 3 in. pipe connecting with the main leading to the sumps. The cement floor is constructed so as to convey all drainage from the extractors to a small well.

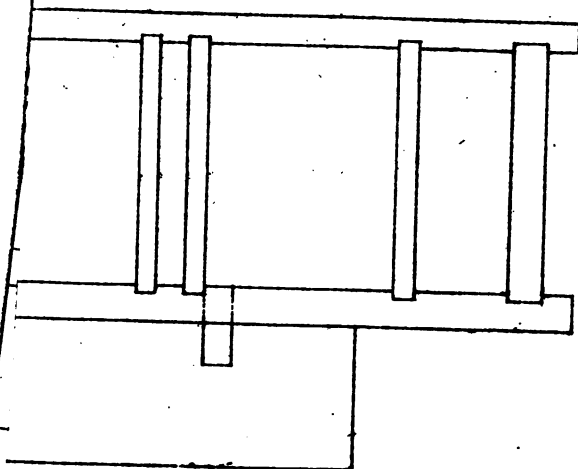
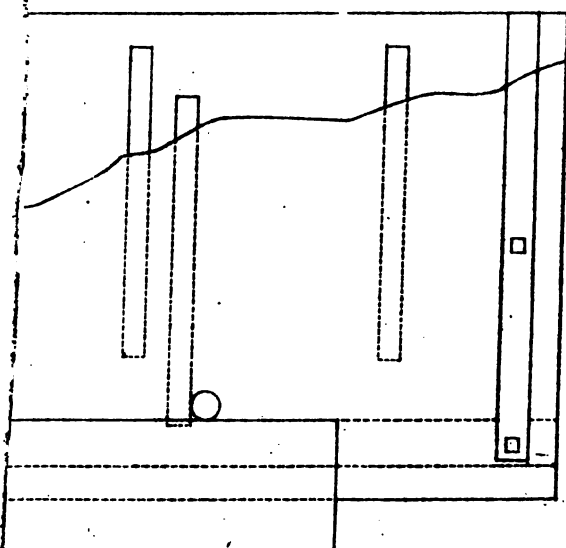
The Roasting Furnace.—In Australia and New Zealand in small cyanide plants this consists of a shallow cast-iron pan or plate, built over a small furnace. A funnel-shaped hood of light wrought-iron is placed over the furnace to carry off the zinc oxide and other fumes.

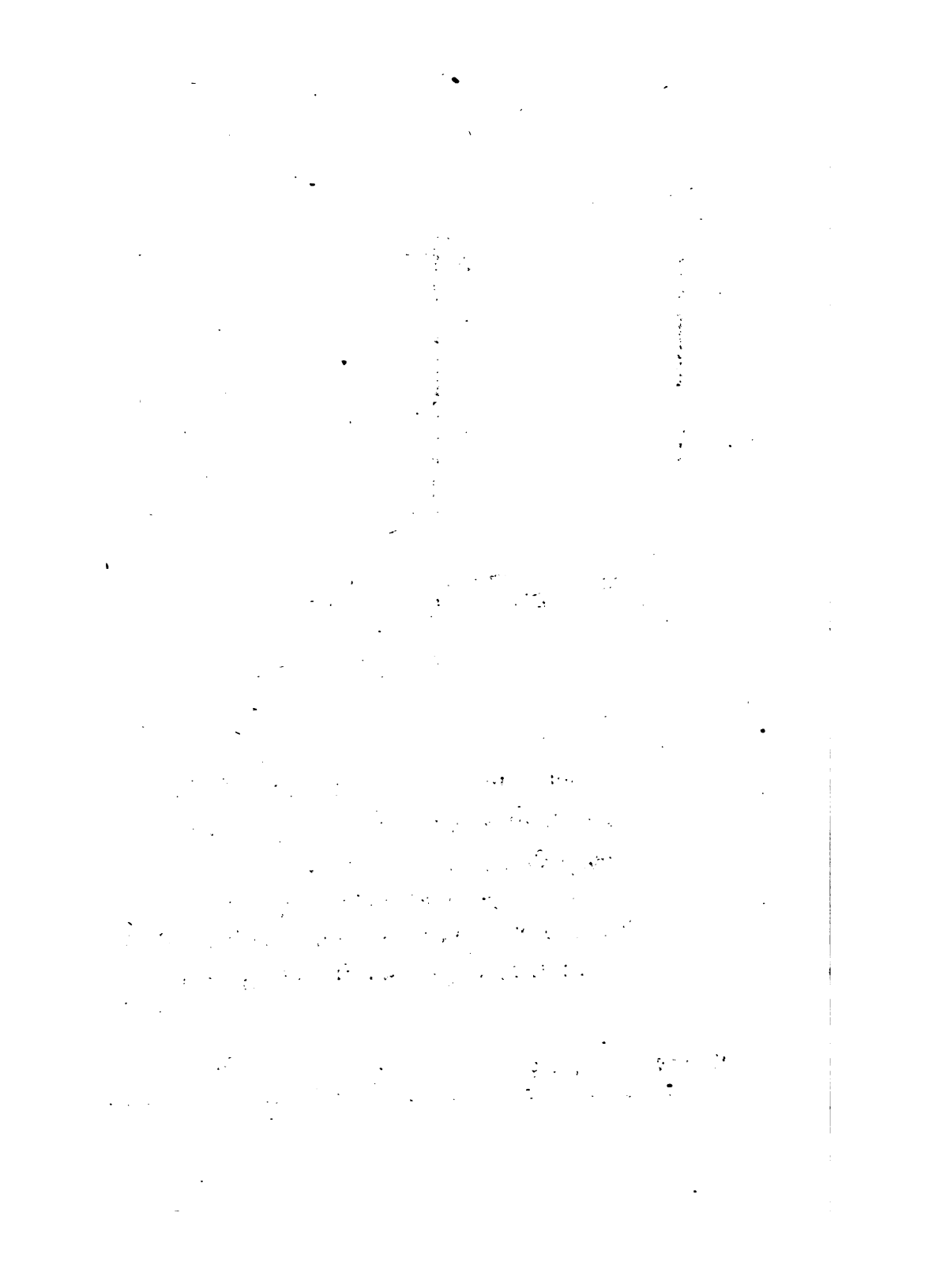
The hood is suspended by a chain or steel-wire rope passing over a pulley overhead, and is balanced by a weight on the end of the chain. The upper or narrow end of the hood telescopes for a few feet into the pipe leading to the condensing flue, and by means of the balance weight the hood can be lowered or raised over the roasting pan or plate as required.

The first part of the condensing flue is a nearly horizontal length of iron-pipe, on which a stream of cold water is sometimes allowed to play, so as to condense the mercury vapours and zinc fumes.

In large cyanide works, a built-in muffle-like roasting furnace

Plate IV





constructed of fire-brick is commonly used. It possesses the advantages that a better regulation of the heat can be effected, and that with care the loss through dusting is less than with open pans.

A roasting furnace capable of taking 50 lbs. of precipitate at a time is shown in Plate V., figs. 1 and 2.

Fig. 3, Plate V., shows a special furnace for roasting slimes. It is a combined reverberatory and muffle furnace. There is a fire-box at one end, a cast-iron hearth in the middle, on which the material lies, and an exit-flue at the other end. The fire-bridge is built in two parts, with a passage between them leading to flues below the cast-iron plate, so that part of the heat is deflected and passes under the plate, while the remainder passes over it with a plentiful supply of air, thus effecting the oxidation of nearly all the zinc.

The Filter-Press or Vat.—In large works, where hundreds of pounds of precipitates have to be handled in the course of a week, a filter-vat with a strong close canvas webbing is found very convenient for washing and drying. The false bottom of the vat is connected with the air-pump. In plants where the slimes are acid treated, a small filter-press is used for washing and drying.

Tell-Tales.—These should be provided for the dissolving and solution tanks, and the storage sumps. They should be marked in feet and inches, so that the workmen can draw off a given depth of solution without any calculation. The cubic content of an inch of depth of each vat or tank in pounds should be marked on a board, so that when so many pounds of solution are required, their equivalent in inches of depth can be ascertained at a glance, thus saving delay and error from needless calculations.

Solution Pump.—This is generally a small centrifugal pump, with a 2 in., 3 in. or 4 in. discharge-pipe. It is driven off the shaft which conveys power to the air-pump.

The Air-Lift for Sands, Slimes, Solutions, etc.—On sloping ground the milling plant and cyanide works can generally be so arranged that the sands and slimes find their way to the leaching vats by gravitation. In cases where natural slopes are not available for mill sites, the battery sands have to be lifted to the leaching vats by artificial means. In South Africa, bucket-wheels; in America, bucket-wheels and sand-pumps; and in Western Australia, bucket-wheels sand-pumps and air-lifts, are commonly employed to lift the tailings.

The air-lift was first introduced and worked on a large scale, some four years ago, by Mr. J. W. Archibald, manager of the Mount Malcolm Proprietary Gold Mines, Limited, at Murrin-Murrin, in Western Australia. Since then it has come into

favour in Kalgoorlie, and seems likely in time to supersede both tailings wheels and sand-pumps in Western Australia.

So far the air-lift has not shown a high efficiency for the power employed ; but it possesses several special advantages which seem to recommend it to mine owners, namely : cheapness of erection, continuous operation, and little wear and tear, having no complicated parts to get out of order. The Associated Mine has two in operation ; Kalgoorlie, six ; Boulder No. 1, four ; Westralia Mount Morgans, two ; Lake View Consols, one ; besides many others.

Judging from the success which has attended the use of air-lifts in Western Australia, it is certain that this form of elevator is deserving of more general application, and for that reason I append the following lucid description of their construction, for which I am indebted to the courtesy of Mr. Max von Bernewitz of Kalgoorlie.

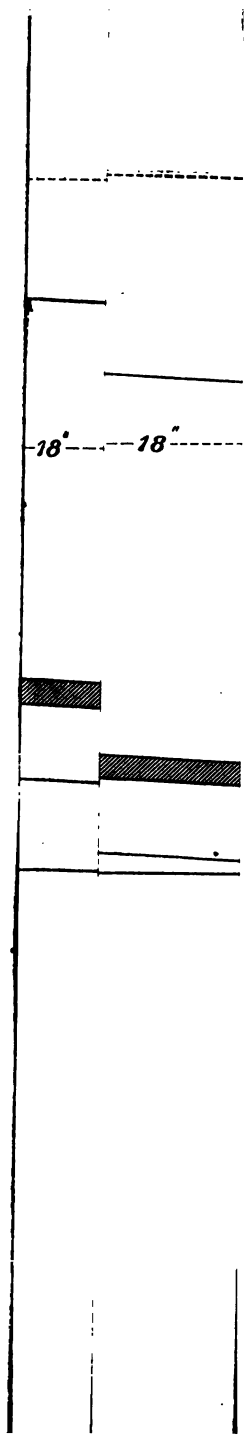
He says : " In erecting an air-lift the first work is that of sinking a well-hole the same depth, or better still, 2 ft. more than the height the pulp has to be lifted. The size of the hole depends on the size and number of lifts to be erected : for example, if one 6 in. lift is to be put up, a well-hole not less than 3 ft. in diameter should be sunk. The reason why a hole is sunk is that the lift has always in it a column of water and pulp, equal to height to be lifted, whereby the air meets with the necessary resistance required to start the column in motion.

" Next, a large pipe, say 2 ft. 6 in. in diameter, with a tapered bottom and of the same depth as the well-hole, is lowered into the latter. The delivery pipe or lift is then fitted together for the correct length, and lowered into the large pipe. Finally, the air-pipe is fitted together and run down at the side of the lift-pipe. At the bottom of the air-pipe is a bend which looks up into the lift about 2 ins. It is desirable to have flanged joints on all the pipes, in order that they may be quickly and easily disconnected in case of an accident.

" At the top of the lift-pipe, a bend may be fitted on, but the practice now is to bolt a launder directly on to a flange at the top of the pipe. By this means the air, as it reaches the top, can escape at once."

It has also been found that the larger the air- and lift-pipes, the better is efficiency attained ; also, that air-lifts should be in duplicate in case of mishap.

If the air is taken from a receiver at, say, five atmospheres pressure, a pressure several times greater than actually required by a lift, and expanded before doing its work, it is found that there is a loss of power. But if the air is taken direct from the compressor



to the air-lift, a far greater efficiency is obtained, as the rising and falling of the level of the pulp acts as a perfect governor to the compressor. The practice on most of the mines is to take air from a receiver at, say, five atmospheres pressure, and conduct it to a lift through a reducing valve; but this is unsatisfactory, and a separate compressor giving large quantities of air at a low pressure is the more economical method.

Again, when a compressor supplies air for many purposes the working of the lift is greatly interfered with. Take, for example, the mill at the Kalgoorlie Mine. Here a compressor running at a high speed supplies air to actuate two solution pumps; also air to agitate slimes; to force the agitated slimes into filter-presses; to dry the presses and work three air-lifts, of which one lifts pulp 20 feet, one sands 30 feet, and one clear water, 15 feet high. All the above require air at different pressures, and interfere with the working of the air-lifts. It is certain that a small compressor for the lifts alone would be more satisfactory.

On the W. A. goldfields, where fuel, water, and wage costs are high, the working of air-lifts is fairly expensive, but in countries where compressors can be driven by water-power with little or no attendance, the cost should not be high.

In the Kalgoorlie Mill a little trouble was experienced with a 20 ft. lift, which elevates red hot roasted ore previously mixed with dense salt water. This lift elevates 700 tons of pulp per diem with an air pressure of 0·7 atmosphere. The pulp when raised is almost at boiling point, and a hard deposit quickly sets around the pipes, and gradually chokes them up at the bottom. This is due to the deposition of lime and magnesia from the ore.

In this mill a proportion of six parts of water to one part of ore was found necessary. The above lift-pipe was 8 in. in diameter, and air-pipe $1\frac{1}{2}$ in.

In the Hainault Mill, a 25 ft. lift has worked for over twelve months with no stoppages, whereas the sand-pump was always hung up for repairs.

At the Associated and Westralia Mount Morgan's mills, lifts are in operation elevating thick slimes from settlers to the agitators. At the latter mill there is a lift which raises pulp from 30 head of stamps, crushing 120 tons per diem. Its dimensions are:—

Height of lift-pipe above well, about	20 feet.
Depth " " below top of well	22 "
Total length of lift-pipe	42 "
Diameter of " "	6 inches.
" air-pipe	$1\frac{1}{2}$ "
Air-pressure	0·8 atmosphere.

At Kalgoorlie, air-lifts are employed to circulate cyanide solution from the bottom to the top of leaching vats, and to raise solutions from sumps. At the Boulder No. 1 Mill, two air-lifts are for lifting mercury in connection with the Reicken process, having the following dimensions:—

Height of lift-pipe, about	12 feet.
Depth of well	4 „
Diameter of pipe forming well	3 inches.
„ lift-pipe	1 inch.
„ air-pipe	$\frac{1}{2}$ „

Air-pressure, 3 to 5 atmospheres.

The size of the plant will depend on the intended capacity per month and the condition of the material to be treated. A plant capable of treating 2000 tons of freely-percolating tailings per month would probably not be able to treat more than half that quantity of dry-crushed ore or slimy tailings.

With dry-crushed ores the depth of the charge seldom exceeds 4 ft., while with sharp tailings the depth sometimes reaches 10 ft. or 12 ft. Cyanide plants for the treatment of dry-crushed ore are therefore provided with a large number of shallow vats; and those for tailings with a smaller number of deep vats.

The following gives the number of leaching vats required for the treatment of different quantities of dry-crushed ore and tailings.

FOR DRY ORE OR SLIMY TAILINGS.—Size of vat: $22\frac{1}{2}$ ft. diameter, 4 ft. deep; charge 30 tons.

2 vats	350 tons per month.
4 „	700 „ „
6 „	1050 „ „
8 „	1400 „ „
10 „	1750 „ „
12 „	2100 „ „

FOR FREELY-PERCOLATING TAILINGS.—Depth of vat, 10 ft.; charge 100 tons.

2 vats	1200 tons per month.
4 „	2400 „ „
6 „	3600 „ „
8 „	4800 „ „

In small plants it is advisable to have a spare leaching vat in case of a temporary breakdown in one of the vats in use.

COST OF PLANT.

The cost of the cyanide plant depends largely on the locality, the material used in the construction, and, to a certain extent, on the condition of the material to be treated. For example, a plant to treat 2000 tons of dry-crushed ore per month will cost more than a plant to treat 2000 tons of tailings.

The approximate cost of plants of different capacities in Australia and New Zealand, when wood is the material used in the construction of the vats, foundations, and buildings, is given below :—

FOR DRY-CRUSHED ORE—

Monthly Capacity.	Cost.
350 tons	£1200
700 „	1500
1050 „	1850
1400 „	2000
1700 „	2400
2000 „	2900

FOR TAILINGS—

Monthly Capacity.	Cost.
1200 tons	£2000
2400 „	2800
3600 „	3500
4800 „	4000

The above estimates include the cost of the laboratory, assay, and melting furnaces, and all the appliances for a successful cyanide plant, but does not include the cost of a bucket tailings wheel or sand-pump for elevating sands, etc.

The cost of steel or iron vats is about the same as that of wood. The steel leaching vats at the Waihi-Silverton, 16 ft. diameter and 4 ft. deep, with central discharge, cost £56 each, and the wood foundations about £10 each.

At Johannesburg the cost of a cyanide plant is about 25s. per ton of tailings to be treated per month. Thus a plant to treat 3000 tons of tailings per month would cost about £4000, not including the cost of tailings wheel or assay and smelting offices. For larger plants the cost is smaller in proportion. The Rand cyanide plants, it should be noted, are not roofed over as they are in many other countries.

In America steel leaching vats cost about 3½ cents per lb. free

on board cars. Thus a 100 ton vat weighing 9400 lbs. would cost about £67. For red wood vats the cost is as follows:—

100 tons	£42 F.O.B.
200 „	80 „
500 „	120 „

The American red wood vats are stated to give every satisfaction. They are generally coated with P. and B. Paraffin paint, of which 40 gals., costing £8, is sufficient to coat a 500 ton vat twice (that is the inside only), two coats on the sides and three on the bottom.

CHAPTER VI.

THE ACTUAL EXTRACTION BY CYANIDE.

Synopsis of the Process.—No hard and fast rules can be followed in the working of the cyanide process, as modifications have to be introduced to meet the special requirements of the different classes of ore; but the essential features of the process, whether for the treatment of ores or tailings, are practically the same in all cyanide plants.

The first operation is the filling of the vats. In the direct treatment of dry-crushed ore, the pulverized material is filled in to a depth of three or more feet, according to its fineness. With a uniform granular product, five or even six feet of ore may be filled into the vat, and, in the case of freely-percolating tailings, the vats may be safely charged to a depth of from six to ten feet.

With tailings it is the practice to fill the vat within 20 inches of the top, because the strong solution or preliminary wash-water when applied causes the charge to shrink several inches.

In the case of acid ores or tailings, a preliminary treatment with a caustic alkali, or at least a water-wash, is necessary in order to remove, or neutralize, the mineral salts and acids which decompose the cyanide solutions.

Oxidizing agents such as barium peroxide, potassium ferricyanide, sodium peroxide, and cyanogen bromide, have been tried in America, South Africa, and New Zealand. The results obtained on a working scale with the first and last of these are very satisfactory.

The strong solution, containing generally from 0.2 per cent. to 0.5 per cent. of potassium cyanide, is then applied. The proper strength of solution must be determined experimentally, and with most ores it will be found advisable to use a larger bulk of solution rather than to unduly increase the strength. Silver ores require a larger bulk of solution than gold ores in order to obtain a fair extraction.

In the case of a dry-crushed ore the solution is allowed to permeate the mass from below, but in the case of tailings it is allowed to flow in over the surface, and soak downwards. The filling of the strong solution generally takes from two to four hours. With tailings it is allowed to stand in contact for twelve hours, after which percolation is begun. With dry-crushed ores slow artificial percolation is commenced at once.

The leaching with the strong solution takes from twenty-four to forty-eight or more hours, according to the character of the bullion; but in the case of ores containing a proportion of fairly coarse gold, it is customary to make the strong solution up to the working strength and pass it through the charge till an adequate extraction is obtained.

The weak cyanide solution, often called the *first wash*, is then applied. It is pumped from the strong sump, and generally varies in strength from 0.1 per cent. to 0.25 per cent. of potassium cyanide. It is allowed to percolate as rapidly as possible, the filtration being facilitated, when necessary, by means of an artificial vacuum, which may be created by a steam exhaust, or an air-pump connected with a vacuum-cylinder.

The first weak cyanide wash is succeeded by two or three washes of solution from the weak sump, containing from 0.02 per cent. to 0.1 per cent. of KCy.

A final washing of clean water is then applied, which serves to displace the preceding weak cyanide wash. By this means the quantity of cyanide solution in circulation remains much about the same.

In the treatment of pyritic ores it is sometimes advantageous to apply a weak cyanide solution before the strong solution. By this means a larger extraction and a saving of cyanide is effected.

The quantity of the strong solution used is about one-third of the weight of the ore. The cyanide and water-washes are each about one-sixth of the weight of the ore.

During the treatment of slimy tailings it is found advantageous to sometimes turn the material over by hand-labour, and thereby effect a more complete washing out of the cyanide solutions from the slimes, which always have a tendency to entangle and retain them or even to transfer it from one vat to another and subject to a second leaching.

Discrepancies between the theoretical and actual extractions are often a source of much annoyance and perplexity to the metallurgist, but in most cases they will be found to be due either to imperfect sampling or incorrect tonnage. At most mines a ton weighs considerably over 2240 lbs.

At the Witwatersrand Goldfields, in South Africa, the tailings

are sometimes subjected to a double treatment—*i.e.*, after the first cyanide treatment the residues are charged into other leaching vats, and again treated with cyanide. A higher extraction will result, but it is doubtful if the difference would cover the cost of the extra handling and cyanide.

The average extraction on the Rand where only high-priced white labour was available, amounts to about 72 per cent., and with double treatment this is raised to 85 per cent. With tailings of fair value the extra recovery is, however, said to leave a margin of profit.

The sequence in which the different operations are applied may be tabulated as follows :—

1. Filling the leaching vats.
2. The preliminary treatment with water or alkaline washes if necessary.
3. Leaching with the strong solution containing from 0·3 per cent. to 0·6 per cent. of KCy.
4. First wash, with cyanide from strong sump, containing from 0·1 per cent. to 0·26 per cent. KCy.
5. Second wash, with cyanide from weak sump, containing from 0·02 per cent. to 0·1 per cent. KCy.
6. Third wash, same as second.
7. Fourth wash, same as last.
8. Fifth wash, with clean water.

The gold and silver in the ore are dissolved by the strong solution, and removed or carried out by the first and second washes. The cyanide solutions are allowed to flow through the zinc extractors, the strong solutions through the strong box, and the weak solutions through the weak box. The first two washes, which generally carry most of the dissolved bullion, are conducted through the strong extractor.

FIRST STAGE.

Filling the Vats with Dry-Crushed Ore.—In the case of dry-crushed ores, the charging of the vats is a simple operation, the only disadvantage being the clouds of dust, which seem to be inseparable from the handling of dry-pulverized ores. At Waihi 90-stamp mill the vats were generally charged by trucks running on to a traveller, provided with hand-traversing gear so as to enable the sand to be tipped in different parts of the vat. In order to prevent packing, the sand was discharged from the trucks on to the platform of a small traveller fixed below the main traveller. By this means the pulverized material was dispersed in a gentle shower over the whole vat.

At some plants the dry pulp is conveyed from the mill to the vats by screw-conveyors or by belt-conveyors, both very efficient methods of transport.

At the Kapai-Vermont Cyanide Works, Kuaotunu, N.Z., the vats were filled directly from the dust-bin, which was situated overhead in an elevated position. By means of a chute, provided with a universal canvas joint, the material was evenly spread over the vats.

When the vat was charged the surface of the ore was made smooth by passing a wide, shallow wooden hoe or rake over it.

This method of filling possesses many advantages over hand-filling by trucks. It is cheaper, more expeditious, and healthier for the workmen, as it raises less dust.

Filling the Vats with Tailings.—One of the most serious disadvantages of wet-crushing with the stamp-mill is the production of fine slimes. All ores, even the most silicious, form a proportion of slimes when wet-stamped; and when clayey or earthy matter, country-rock, iron or manganese oxides are associated with the ore, the proportion of slimes is often excessive.

In many places the slimes are very valuable, sometimes even more so than the sands. In leaching processes they prove very refractory, as they seriously interfere with the percolation and washing, thereby causing the extraction to be both costly and imperfect. When the slimes are irregularly distributed through the sands the cyanide solutions form channels through them, and imperfect leaching is the natural result.

At the Witwatersrand Goldfields, where wet-crushing, followed by copper-plate amalgamation, is at present universal, there are two methods in use for dealing with the tailings before treatment with cyanide. The pulp from the copper-plates is lifted by a bucket wheel, run through a launder, and then classified by *spitzkasten* or *spitzluten* into two products, namely:—

(a.) Sands and slimes, 80 per cent.

(b.) Concentrates containing some sands, 20 per cent.

The concentrates are collected in a storage tank, and kept under water to prevent oxidation, while the sands and slimes pass on to the cyanide vats for treatment, either by what is known as the "Direct Filling" process or by the "Intermediate Filling" process.

At the extensive works of the Langlaagte Estate Gold Mining Company, the tailings, after leaving the plates, are concentrated, and then run into three large settling dams, each capable of holding 7000 tons. The sands settle in the dam, while the slimes are carried off by the overflow, and allowed to run away. The tailings, now free from slimes, are hauled from the dams in trucks by means of two endless steel-wire ropes, and run up to an overhead

tram-line, from which they are dumped into the leaching vats ready for treatment.

Intermediate Filling.—At the Robinson, Princess, and many other mines at Johannesburg, the tailings, after being subjected to concentration, are run into intermediate settling-tanks, which are circular in shape, and sufficiently large and numerous for the requirements of the mill. At the Robinson plant, treating 330 tons per day, there are six circular wooden vats, each 24 ft. in diameter, and 11 ft. in depth. This gives a settling surface of 32 square feet per stamp.

From these intermediate settling vats the tailings are distributed to the leaching vats. When the slope of the ground permits it, the settling vats are placed above the leaching vats, the tailings being discharged from holes in the bottom; but when they are below the level of the leaching vats, the tailings are hauled up in trucks actuated by endless wire ropes.

To ensure a fairly even distribution of the sand and slimes in the leaching vats, a simple and ingenious automatic distributor, invented by Messrs. Butters & Mein, is now in use in all parts of the world. It consists of a central casting, with a vertical spindle revolving in a foot-step, and carrying a conical hopper at the top, from which radiate twelve or sixteen iron pipes with bent ends. These pipes vary in size from $1\frac{1}{2}$ in. to $2\frac{1}{2}$ in. in diameter. To their discharge ends are attached flattened nozzles to assist in spreading the tailings over a wide area. A coarse screen is placed over the central hopper, or bowl, so as to prevent stones or pieces of wood, or other *débris*, from choking the pipes. The distributor is fixed on an iron column in the centre of the vat, and the reaction of the pulp escaping from the bent pipes causes it to slowly revolve in a manner similar to that of an ordinary garden sprinkler. (Fig. 4, Plate V., and Plate VI.)

To insure success, the collecting vat must be filled with clean water before admitting the pulp, otherwise the slimes would settle with the sand until the overflow of water began. While the machine is running, there must be a continual overflow from the vat to carry off the fine slimes. The discharge, or overflow, from the settling vat takes place at all points of the circumference, being received into an annular ring, surrounding the top of the vat, which conveys it to the slime-pit. The overflow of slimes must be continuous until the vat is full of sand, therefore, when the battery stops, a regular quantity of water should still be supplied to the vat.

The vats are provided with filters, and when full of tailings the water is allowed to drain off for fifteen or twenty-four hours, while six hours before discharging, holes are dug down to the discharge

doors to let more water flow out. When drained, the tailings are discharged into the leaching vat below, through "bottom-discharge" doors, or into trucks, which are then run into the tank house, where their contents are emptied into the leaching vats. At the same time the required amount of lime (quicklime ground in a ball-mill) is added in such a way that it gets distributed through the whole mass.

The advantages claimed for intermediate filling are:—

- (a.) By means of the distributor, nearly all the sands are collected in the intermediate tanks, the bulk of the slimes escaping. (The quantity of slimes depends on nature of ore, size of screen, and size of battery-box.)
- (b.) When discharging through the bottom, the sands during the operation get thoroughly mixed, thus being in the best condition for treatment.
- (c.) Oxidation of pyrites is very slight, so that little cyanide will be consumed.
- (d.) A higher extraction can be obtained owing to the presence of the fine sands, from which 85 per cent. of the gold contents can be extracted by cyanide.
- (e.) The cost of treatment is very small.

The principal disadvantage is the tendency of the distributor to form deposits of slime, through irregular distribution, at places on, or near, the bottom of the tank, thereby causing some trouble in draining off the water.

Direct Filling.—This method is in use at the works of the City and Suburban, Crown Reef, and other companies at the Witwatersrand, and consists in conducting the pulp, as it leaves the copper-plates, into a classifier or *spitzlute*. In this the pulp is divided into two streams: one, the overflow, carrying slimes and fine sands; the other, carrying the coarse sands, together with some fine sand and slimes, which are conveyed by an india-rubber hose to the leaching vat, where they are distributed by moving the hose over the whole area of the vat. The excess of water is carried off by adjustable gates fitted inside the vat, taking with it some fine sand and slimes.

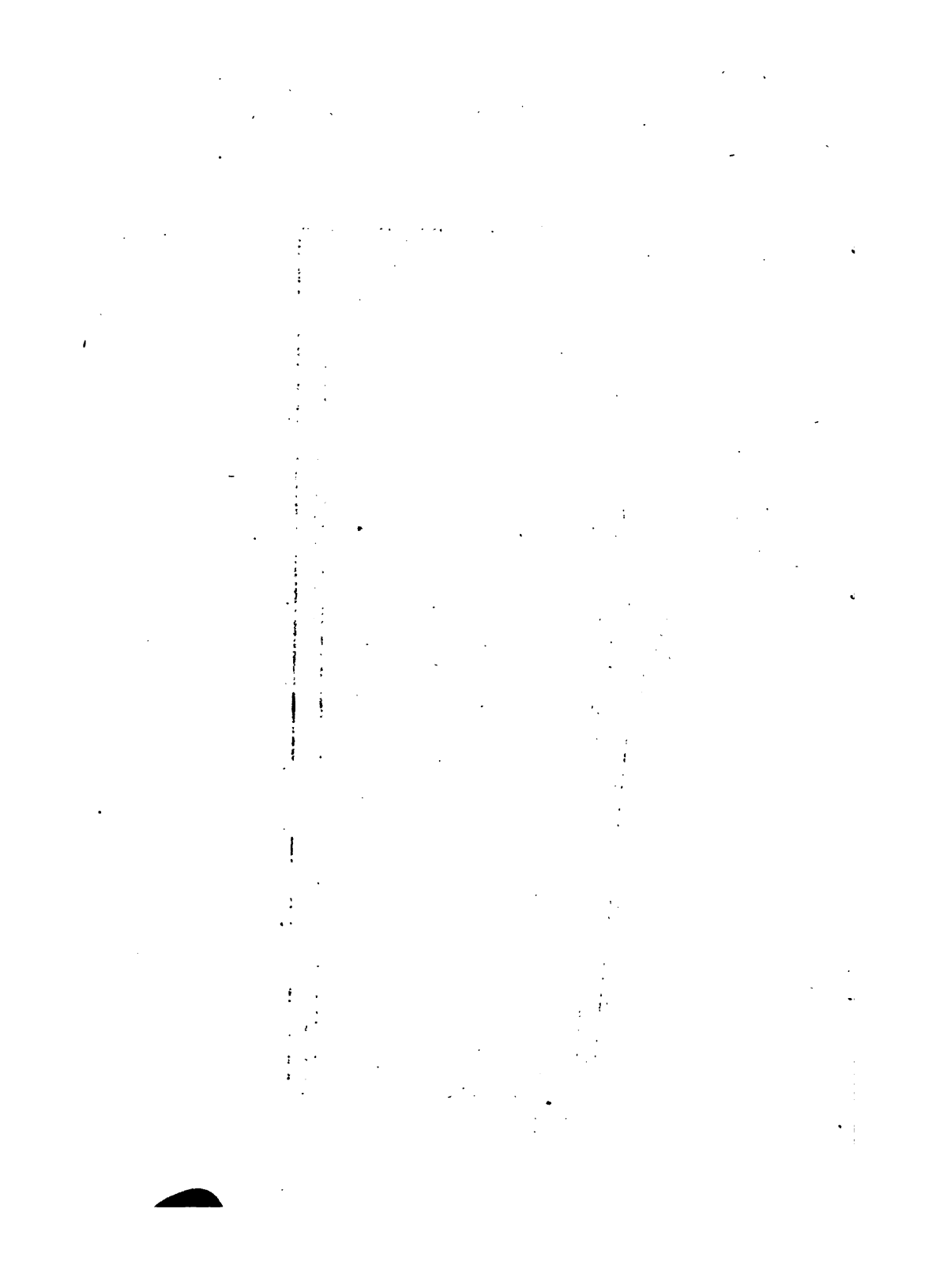
The advantages claimed for direct filling are:—

- 1. That pyritic tailings are exposed to the minimum of oxidation.
- 2. A second handling of the tailings is avoided.
- 3. A rough preliminary classification is effected, thus separating the fine slimes.

The principal disadvantages of this method are:—

- 1. The packing of the tailings, which prevents a complete draining of the contained water.

INCHE



2. The unequal distribution of the sands and slimes, which militates against a perfect extraction by favouring the formation of channels in the mass during the subsequent leaching.

The method of classifying the pulp by a *spitzlute*, and direct filling, has been in operation at the Great Mercury Cyanide Works, Kuaotunu, N.Z., since 1892. When the leaching vat is charged, the water is drawn off through the filter-bed, and after the tailings have drained they are turned over by hand so as to loosen them, and at the same time thoroughly mix any slimes present with the sands.

SECOND STAGE.

The Preliminary Water or Alkaline Wash.—This treatment is only necessary in the case of ores or tailings containing the decomposition products of pyrites, or other base sulphides.

The drying of a pyritic ore in a kiln for dry-crushing and direct treatment generally results in the formation of soluble sulphates, which are destructive to cyanide. At the works of the Woodstock Gold Mining Company, at Karangahake, N.Z., the consumption of cyanide from this cause was very heavy; but the preliminary treatment with a caustic alkali, according to the later report of the management, has effected a great saving of cyanide, accompanied by a higher extraction.

The products of the partial oxidation of iron pyrites in tailings are principally free sulphuric acid, soluble sulphates, and insoluble basic sulphates, all of which cause a large consumption of cyanide, and must, therefore, be removed or neutralized before cyanide treatment is commenced.

If the tailings are very acid, and a considerable proportion of the salts are found to be soluble in water, the general practice is to apply a preliminary water-wash. In order to neutralize the remaining acid, a quantity of the solution of caustic soda, equal to about 1 lb. of the salt to every ton of ore, is allowed to stand in contact with the tailings for two hours, and then drained off into the alkali-sump.

The necessary quantity of caustic soda can be determined experimentally by the method recommended by Feldtmann. (See Chapter IV.)

At the Witwatersrand Goldfields powdered caustic-lime is generally used in place of soda. With very acid tailings as much as $2\frac{1}{2}$ lbs. to the ton is used. It is applied by adding the requisite amount to each truck-load before being dumped into the leaching vats.

The author generally used lime at the Government Experimental Works at the Thames, and found it preferable to caustic soda, as it is not attended with the production of ferrocyanide of zinc, which fouls the zinc in the extractor. On the other hand, it must be remembered that the addition of lime only improves the extraction when sulphides are not present in the material being treated.

When the cyanide contains sulphide, the extraction is certain to be slow and unsatisfactory. The sprinkling of lead acetate on the top of the tank will precipitate the sulphur of the sulphide as lead sulphide, thus leaving the cyanide free from sulphide.

In the case of very acid tailings, Feldtmann strongly condemns the practice of conducting the preliminary washing in the leaching vat, on account of the possibility of the acid acting on the residual cyanide in the vat, and thereby liberating sufficient hydrocyanic acid to dissolve an appreciable quantity of gold, which he says would be lost, as it is not precipitated by zinc.

He suggests that the washing should be effected in one vat, and the leaching in another; and considers that the extra costs of handling would be more than made up by the higher extraction.

It is doubtful if hydrocyanic acid has any action on gold; but it is probable that the ascending acid by uniting with any alkali present would form an alkaline cyanide which would at once react on any gold to which it had access.

THIRD STAGE.

Strong Solution Leaching.—With dry-crushed ore, the strong solution, about *one-third* the weight of the charge, and generally containing from 0·2 per cent. to 0·5 per cent. of available potassium cyanide, is allowed to pass into the vat from below. When the solution stands *two inches* above the surface of the ore the stop-cock is shut, and the solution lying below the filter is then drawn off, and filtration commenced. The strong solution generally takes from twenty-four to thirty-six hours to percolate through.

In the case of tailings, the strong solution is added on top. After standing twelve hours, to allow the solution to permeate all the lumps of slimy material, the solution is slowly drained off, and passed through the zinc extractor. Small quantities are occasionally drawn off to assist diffusion of the solution through the mass.

It is a very common practice to apply a weak solution from the sump containing, say, 0·05 to 0·10 before applying the strong solution, the intention being to allow the acids and foreign matters

destructive to cyanide to expend themselves on the weak solution, before applying the strong solution.

When the ore contains a large proportion of silver—say, from five to eight parts to every part of gold—it will be found necessary to adopt one of two courses in order to obtain a satisfactory extraction.

Either a much greater bulk of potassium cyanide solution must be used to leach the ore—say, a quantity equal in weight to that of the ore taken—or else a much stronger solution must be used.

The former, only, of these two courses would be applicable if the ore contained even a small proportion of copper oxide, carbonate, or sulphide, or of sulphide of antimony, since the solubility of these in all solutions of cyanide, but especially in the strong, would render a fair extraction impossible, besides causing a heavy loss of cyanogen.

The sulphide ores of silver are more slowly soluble in cyanide than gold, and for this reason the treatment of such is always more expensive than that of ordinary tailings or gold-bearing ores. On the other hand, the chloride of silver (kerate or horn silver) is more readily acted on than gold, the extraction generally exceeding 80 per cent.

In the practical treatment of ores and tailings by cyanide, one of the first anomalies to attract the notice of the metallurgist is the fact that the strong solution, while it 'loosens' the gold, so to speak, yet does not carry it away, this being effected by the first and second cyanide washes.

It is found that the first portions of the strong solution, draining from the charge, contain only from 0.02 per cent. to 0.1 per cent. of cyanide; but the remaining portions come off stronger and stronger after a lapse of eight or twelve hours, until, towards the end of the strong leaching, the maximum strength is reached, after which the strength declines a little before the application of the first cyanide wash.

The first portions of the solution during the strong cyanide leaching, being weak in cyanide and low in gold, are, therefore, passed through the weak zinc extractor, while the later portions, together with the first and second cyanide washes, are conducted to the strong extractor.

It will be found in practice that fresh solutions are more active in dissolving gold than used solutions which in passing through the zinc-extractors become charged with the inert compound zinc-potassic-cyanide.

Strong Sump-Solution Wash.—After the strong solution has completely drained off, the strong sump-solution is applied

from above, being run in on the top, or surface, of the ore. Its strength varies from 0.1 per cent. to 0.25 per cent. of KCy, and the quantity applied from one-third to one-quarter of the weight of the ore.

This weak solution is sometimes allowed to slowly percolate through the charge, but more often it is drawn off as rapidly as possible, for the more rapidly the wash solutions are drawn off the more effective is the washing. The filtration is assisted by opening the stop-cock connecting the bottom of the leaching vat with the vacuum-cylinder.

The percolation of the weak solution generally takes from twelve to twenty hours, the time depending on the condition of the pulp forming the charge.

Weak Cyanide and Water Washes.—The number and strength of these will depend entirely on the character of the ore, or tailings, being operated on. In some cases it is found necessary to apply as many as three or four weak cyanide washes, from the weak sump, and then a final water wash; in other cases the whole treatment consists of the strong cyanide leaching followed by a cyanide wash from the strong sump, a cyanide wash from the weak sump, and one or two final water washes.

The quantity of each wash is in most cases one-half that of strong cyanide solution.

The effect of the different washes should be carefully determined by assaying the residues after each washing, and also the wash solutions as they drain from the vat. By this means the necessary number of washings will soon be ascertained.

The assay sample should represent a fair average of the charge in the vat, and is easily and reliably obtained by taking a large number of cores, the full depth of the charge, by means of a tube shaped something like a cheese sampler. The scores should be dried and then sampled down for assay.

Briefly summarized, the aim of the different solutions which have been applied to the material in the leaching vat is as follows:—

- (a.) Alkaline wash, to neutralize acidity so as to prevent waste of cyanide.
- (b.) Strong solution, to effect solution of gold.
- (c.) Weak solutions and water washes, to displace cyanide solution and to prevent cyanide from being thrown away with the residues.

The different operations to be undertaken in the dry-crushing and direct method of treatment may be summarized as follows:—

1. Dry crushing.
2. Cyanide treatment.

3. Copper-plate amalgamation of tailings.
4. Concentration of tailings from plates.
5. Treatment of concentrates.

With wet-crushing the different operations are :—

1. Wet-crushing.
2. Copper-plate amalgamation.
3. Concentration.
4. Treatment of concentrates, by cyanide or other means.
5. Treatment of tailings by cyanide.
6. Treatment of slimes by cyanide.

REMARK.—When leaching slimy sands the successive solutions and washes should only be partially drained, just sufficient to allow assay samples to be taken. If very fine sands, or sands containing slimes, be thoroughly drained, they become almost impervious to subsequent solutions. Hence it is important to partially drain so as to leave a cushion or jacket of solution round each particle of sand, the said jacket preventing packing and thereby permitting its easy replacement by the solution following.

CHAPTER VII.

THE TREATMENT OF SLIMES.

WITH all methods of wet-crushing and pulverizing the formation of a certain proportion of slimes seems inevitable, and when the ore contains metallic oxides, clay, or other earthy matter, the production is often very large. In ores containing a certain proportion of very fine or 'float' gold, the slimes are generally a valuable product, and their successful treatment has engaged the attention of metallurgists for many years.

Since the introduction of the cyanide process, many attempts have been made to leach the slimes resulting from wet-crushing, on an economic scale, and the subject is one to which the author has devoted much thought and experiment for a number of years. The problem is principally a mechanical one, consisting in the difficulty of separating the solutions from the slimy mass, rapidly, effectively, and at such a cost as to permit the treatment of slimes of low value.

Up to the present a great many different devices have been tried, with varying degrees of success. Among these may be mentioned compression by hydraulic and other presses; agitation and centrifugal force; agitation and decantation; and agitation with filtration, aided by an artificial vacuum.

The dry-crushing of ores by means of the Californian stamp battery is always attended with the production of so large a proportion of slimes that only a shallow depth of the pulverized material can be leached in the direct method of cyanide treatment.

The stamp battery was invented for wet-crushing, for which it stands unrivalled for all classes of ore. For dry-crushing it is a most unscientific machine, on account of its inability to discharge the dust when reduced to the requisite fineness. Nevertheless, no better has yet been invented.

In New Zealand and Australia every variety of pan and ball-mill, all the best known roll machines, and pulverizers of many

different designs have been tried, but, except where the ore was very soft and friable, and the object is to produce slime, all have been discarded for the stamp, which is the only machine known at present which is able to cope successfully with the hard quartzose ores.

The depth of dry pulverized material placed in the leaching vats seldom exceeds three feet, even with the most favourable silicious ores. This necessitates a large plant to treat a comparatively small output of ore, and a correspondingly greater cost per ton for treatment.

Author's Experiments:—Early in 1893 the author treated a large parcel of ore from the Monowai mine, in the Thames district. The ore consisted of hard bluish and grey-coloured splintery quartz, containing a considerable proportion of sulphides of iron, copper, lead, and zinc.

The ore was dried, dry-crushed, sampled and assayed, showing a value of £5, 5s. per ton. The crushing was effected in a stamp battery, which produced a large quantity of the finest slimes. These slimes rendered it impossible to effect the leaching by percolation, even with the aid of a vacuum. When mixed with water, the slimes, when only 4 in. thick, settled on the filter-cloth, forming an impervious bed, through which it was found impossible to draw off the solution.

The pulp was then subjected to agitation, by which the dissolution of bullion was effected in six or seven hours. The separation of the solution from the pulp, however, was a long and tedious operation, and extended over eight days. It was effected, but not very satisfactorily, by agitating the ore, allowing the slimes to settle, and then drawing the clear solution off by a syphon. The weak solution and wash-waters were added in succession, and the same operation performed after each.

In order to ascertain the degree of fineness to which the ore was reduced when crushed, a number of experiments were made with a 60-mesh, 40-mesh, and an ordinary battery-punched screen, and it was found that the results were, in each case, as follows:—

WITH 60-MESH—

A.	18%	remained on	.	.	.	.	8100	sieve.
B.	5%	"	"	.	.	.	3600	"
C.	100%	passed through a	.	.	.	.	1600	"

WITH 40-MESH—

A.	22%	remained on	.	.	.	.	8100	sieve.
B.	12%	"	"	.	.	.	3600	"
C.	3%	"	"	.	.	.	1600	"
D.	100%	passed through a	.	.	.	.	900	"

WITH PUNCHED-SCREEN, EQUAL TO 30-MESH—

A.	26%	remained on	.	.	.	.	8100	sieve.
B.	18%	"	"	.	.	.	3600	"
C.	13%	"	"	.	.	.	1600	"
D.	4%	"	"	.	.	.	900	"
E.	100%	passed through a	.	.	.	.	625	"

Taking the values of the different products separately, it was found that the finest in all cases gave the highest values. This also received confirmation from the circumstance that the fine dust, which had collected on an elevated platform during crushing, assayed higher than the dry material in the bin.

The relative values were as follows:—

From dust-bin,	.	.	.	.	£5	5	0	per ton.
„ platform,	.	.	.	.	6	13	6	„

The ratio of gold to silver in the pulp from the dust-bin was nearly 1 to 9, and in the dust from platform 1 to 12, thus showing an increase of the silver contents.

In 1893 the author obtained, with two of his assistants, a patent for a combined agitation and leaching process, which may be described as follows:—

The appliances used in the operation consist of a shallow circular vat, a vacuum-cylinder, and an air-pump. The vat is provided with four revolving arms, to which soft rubber brushes are attached. The bottom of the vat is provided with a false bottom, consisting of a wooden grating covered with wool-packing or other webbing. The operation is conducted as follows:—The leaching solution, made up to the required strength, is first conducted into the vat. The revolving arms are then set in motion, and the dry pulp or fine slimes introduced. The agitation is continued for six hours, or until the extraction is complete. A stop-cock, in a pipe connecting the false-bottom of the vat and the vacuum-cylinder, is then opened, and the air-pump started. The effect is immediate. The clear solution at once begins to drain over into the vacuum-cylinder, the brushes on the revolving arms preventing the slimes from settling and choking up the filter-cloth. When the slimes have been drained down to a thick paste, the first wash is added, the pump again started, and the slimes drained as before. The subsequent washes are applied in the same manner, and, when the washing is completed, a plug or door is opened, and the leached slimes are sluiced out. The whole operation of leaching and washing takes from eighteen to twenty-four hours.

This process was adopted by the author for the treatment of

several tons of ore from the Monowai mine, at Waiomo, with complete success. This ore contained a large proportion of clay and iron oxides, and, when dry-pulverized, formed a pulp which defied all the ordinary methods of percolation. Trial tests were also made with parcels of very fine slimes, and in all cases the results were most satisfactory.

The author continued his experiments for the treatment of slimes in another direction, and in 1895, with Mr. G. W. Horn, obtained a patent for an improved leaching process for the treatment of slimes and other fine matters by cyanide or other solvents.

The necessary apparatus consists of a solution vat, an air-compressor, and a leaching vat provided with a filter-frame and webbing, both at the top and bottom. The vat may also be provided with an air-tight cover, so as to permit the creation of an artificial vacuum to facilitate the filtration.

In practice, the leaching vat, which may be constructed of wood, or any suitable material, and of any convenient size, is charged with cyanide solution. The slimes are then introduced and well mixed with the solution, which is then allowed to penetrate and stand in contact with them for two hours. At the end of that time the top filter-frame and webbing is put on, and additional cyanide, or other solution, is allowed to penetrate and permeate the slimes from below, thus displacing the solution already in the vat, and causing it to pass through the top filter-web as a clear liquid, which is conducted away by a launder.

In order to prevent the solution selecting channels through the slimes, or other matter in the vat, during the upward filtration, the material is agitated by the discharge of compressed air through it from distributors placed in the sides and bottom of the vat.

The gold in the slimes, being necessarily very fine, is dissolved very quickly, and the percolation can generally be started two hours after charging the leaching vat. When a portion of the gold is coarser than usual, or when it exists in the form of amalgam, it is found that the agitation caused by the discharge of compressed air during the leaching greatly accelerates the dissolution.

The novel features of the process consist of upward filtration, and agitation by means of discharges of compressed air through the mass.

The results of a number of small tests, on the finest slimes obtainable, have proved most satisfactory, and more extensive trials will soon be undertaken. One air-compressor would be sufficient for a number of leaching vats; and the author is confident that the process is capable of very wide and useful application.

Acting on a suggestion thrown out by Mr. C. Wichmann, the author obtained from Mr. E. G. Banks, metallurgist to the Waihi Gold Mining Company, the results of a number of experiments, showing the relative values of the different degrees of fineness of the pulverized ore. These results are very instructive.

EXPERIMENTS showing degree of fineness and relative values to which "Martha" ore was reduced by stamping through 30-mesh Screens.

		Value per Ton.
A.	0.36% remained on 30-mesh (900 holes)	£4 2 6
B.	2.16% " " 40-mesh (1600 holes)	3 18 10
C.	9.29% " " 60-mesh (3600 holes)	3 11 2
D.	25.72% " " 80-mesh (6400 holes)	4 9 0
E.	74.28% passed " " "	5 7 1

Similar Experiments with 40-mesh Screens.

A.	0.3% remained on 40-mesh (1600 holes)	3 12 7
B.	7.8% " " 60-mesh (3600 holes)	3 7 11
C.	14.7% " " 80-mesh (6400 holes)	3 11 4
D.	85.3% passed " " "	3 19 2

*Experiments showing value of Dust rising from stamp—
Dry-crushing.*

		Value per Ton.
A.	Dust obtained from floor of mill to 10 ft. high	£6 16 4
B.	" " 10 ft. to 20 ft. high	7 2 4
C.	" " 20 ft. to 30 ft. " "	7 3 1
D.	" " 30 ft. to 40 ft. " "	7 19 11
E.	" " 40 ft. to 60 ft. " "	8 13 8
	Average value of ore from which this dust came	4 1 4

The different processes of slime treatment at present in use may be classified as follows :—

1. **Decantation**, as practised in Africa, America and New Zealand.

2. **Filter-press method**, as practised in Western Australia and New Zealand.

3. **Electro-chemical precipitation.**

4. **Sun-drying.**

Decantation.—In 1896 Mr. C. Butters, then manager of the Rand Central Ore Reduction Company, erected a large slime plant at the Robinson mine, at the cost of nearly £60,000. The salient features of the treatment are the classification of the slimes into

three products. The first and second products, which consist of fine sands, are treated by ordinary leaching in vats; while the fine slimes are leached in vats with revolving agitators, by a process patented in New Zealand by the author early in 1893. The agitators are about 10 ft. in diameter, and provided with a filter-frame and webbing. During agitation the solution is drawn off by means of an air-pump connected with a vacuum-cylinder. A fuller description of this process is given further on.

On the Rand, the treatment of slimes has naturally been a very serious problem, to the solution of which much time and money have been devoted. The following method of treating slimes, known as the "natural settlement" process, has been in use at the Crown Reef since August 1896, with satisfactory results. It was first devised by Mr. J. R. Williams, and afterwards adopted by the Robinson Slime Works.* To the water, carrying the slimes from the separator plant, is added milk of lime, the slimes being thus precipitated in a flocculent form. The supply of lime is regulated by an automatic feeder, as too much lime is as bad as too little, since it interferes with the efficient precipitation of the gold.

The slimes are settled in three large pointed boxes, two of them 20 ft. x 20 ft. and 10 ft. deep, and the third 40 ft. x 40 ft. and 10 ft. deep. The settled slimes are drawn off at the bottom and pumped into the first two of eight treatment vats, about 90 per cent. of the water contained in them having been separated.

The vats are each 32 ft. in diameter and 10 ft. deep, having a conical bottom. More water having been separated from the slimes by allowing them to settle, they are sluiced into a pump by a jet of cyanide solution and transferred to a second series of two vats filled with solution, and the strength of which is brought up to 0.1 per cent. of KCy.

About 80 per cent. of the gold is dissolved in the passage through the pumps, but agitation is continued from one to two hours by withdrawing the solution at the bottom and discharging it in oblique jets at the top and through the sides.

The slimes are then allowed to settle, and the clear solution is drawn off through side-cocks (which have been replaced by syphon pipes at the Robinson Works) and passed to the precipitation boxes.

The residual slimes are then pumped successively into the third and fourth series of two vats, where they are further agitated with very dilute solutions of cyanide and allowed to settle. These solutions do not pass to the precipitation boxes, but are transferred to the preceding series of vats.

* *Chem. and Met. Soc. of S. Africa*, July 1897.

The "strong" solution from the second series of vats is run into two settling tanks, 15 ft. in diameter and 5 ft. deep, where it is allowed to clarify. Electrical precipitation is used. From 6643 tons treated, an actual extraction of 60.5 per cent. was obtained, at a cost of 3s. 9d. per ton.

The concentration of the solution as regards its gold contents when decanted, was first successfully introduced by Mr. J. R. Williams at the Crown Reef plant, in order to reduce the amount of solution passing through the precipitating boxes. Naturally, when a given amount of gold in solution is obtained from the slimes per twenty-four hours, if this amount of gold can be concentrated into a small volume of solution, a small precipitating capacity is required. For these reasons Mr. Williams introduced the system of double washing; that is to say, the first solution that is applied to the slimes after decantation is not clarified and passed through the precipitation boxes, but is run into what is called an "intermediate tank," and then pumped up for use again upon a fresh charge of slimes. After this second decantation, it is run through the clarifying apparatus to the boxes for precipitation. The settled slimes are then re-pulped with an equal volume of solution from the precipitated sump; that is, solution which has passed through the precipitation boxes. Theoretically, the enrichment of the unprecipitated solution could be carried on until it was equal to the value of the slimes, but in practice it has been found that the first wash which has not been precipitated, and the second wash which has been precipitated (two washes altogether), make the simplest and most perfect system.

South African Practice.—The main features of the decantation process as practised in South Africa are as follows:—

1. *Spitzlute separation* of sands and slimes.
2. *Spitzkasten concentration* of slimes.
3. *Collection and settlement* of slimes in collecting vats provided with decanting syphon.
4. *Agitation* of slimes with cyanide solution.
5. *Settlement* of slimes and *decantation* of solution for precipitation.

At the Rand mines, the agitation is generally effected by a centrifugal pump, but in some cases revolving stirrers are used.

New Zealand Practice.—The method of slimes treatment adopted by the Waihi Company at their new mill at Waikino in 1901, is based on the best South African practice, differing only in the separation of the gold-containing solutions which are separated from the slimes by filter-presses instead of repeated decantation as at the Rand.

A detailed description of the methods of slime treatment at the

Waihi Company's mills of 320 stamps, furnished by the company's metallurgist, Mr. E. G. Banks, will be found in Chapter XII.

Filter-Press Practice.—This process has been revived in New Zealand during the past few years, but it is in Western Australia that it has received its widest and most successful application. The scarcity of water and the clayey character of the oxidized ores of that tropical country presented a difficult problem to the metallurgist and chemist. Compared with the decantation process, a higher average extraction is being obtained, with a lower consumption of water; but the assay value of the residues is higher than in South Africa, where it has been reduced as low as $9\frac{1}{2}$ grains of gold per ton. A consideration of the practice shows that the higher extractions at Kalgoorlie are due to the higher grade of the slimes treated.

Details of Process.—The following details of the process as practised in Western Australia are extracted from an instructive paper recently published by Mr. Donald Clark, Director of the Bavinsdale Mining School in Victoria.*

The ore at Kalgoorlie, as a rule, is dry crushed, Krupp's ball mills being mainly used; after crushing, it is wholly roasted. Several types of furnaces are on the field, including the shaft furnace (similar to those at Mount Morgan), Brown's, Ropp, Holthoff-Wethey, and Edwards' mechanical furnace. It is noteworthy that the last is most valued, on account of its lowness in cost and its perfection in roasting.

The analysis of typical ore from the Lake View Consols mine is appended:—

Before roasting.		After roasting.	
Insoluble	. 62·80	Insoluble	. 66·50
FeS	. 8·27	Fe ₂ O ₃	. 11·06
CaCO ₃	. 13·89	CaO	. 7·72
MgCO ₃	. 7·37	MgO	. 3·48
		SO ₃	. 6·58
		S (undecomposed)	·11

These results show that the ore is not a complex one, since it consists of the usual gangue and a comparatively small percentage of pyrites. The presence of telluride of gold gave rise to the greatest difficulties in the treatment of this ore.

The method finding most favour is to pass the whole of the roasted ore into a hydraulic separator, the fine gold and slimes flowing over the top, while the coarse gold, the partly roasted sulphides, and the coarse sands are drawn away at the bottom. The slimes are led into spitzkasten, where 50 per cent. of the

* Clark, *The Australian Mining Standard*, Dec. 5, 1901.

water flows off at the top clear, and the mud, containing 50 per cent. of dried slimes, is run into agitators, where it is agitated with cyanide solutions and afterwards filter-pressed. The coarse sands from the hydraulic separator are run over copper plates, then over percussion tables, where the coarser concentrates are extracted. The sand which escapes from these is run into a vat, and there treated in the usual way with cyanide solutions. The concentrates from the table are ground to slimes in an amalgamating pan; the slimes are sent to the agitators, thence to the filter-presses.

The Dehne filter-press is the favourite press in use and has given great satisfaction. The Martin press has lately been placed in the market and is worked as follows:—

The slimes are forced in through the slime valve and passage, and thence find their way through the ports or side openings of the open frames which they fill up, the liquor escaping by the drain cock.

When the water ceases to run, these plates are full of fairly caked slime. The cocks are then closed. Solutions for cyaniding or washing, as the case may be, are then forced in through the solution channels. These find their way into the solution plate, and are forced from the corrugated plate through the perforated one, then through the slime cake, through the filter-cloth, the perforated plate, and down the corrugated surface of air-plate whence they are led away to the zinc boxes. It should be stated that any imprisoned air is first got rid of by opening the air escape cocks, and displacing the air by the solution. When the liquid starts to run through the air-cock, the solution plate is closed.

When cyaniding in presses, the escaping water is clear enough to be led back to the separators; the cyanide solutions are forced through at a pressure of 90 lb. per square inch until the gold is extracted. It takes 90 minutes to fill, 90 minutes to extract the gold, and 90 minutes to empty three such presses. In order to displace any wash water or solution, a current of compressed air is turned on to the solution passage for a few minutes, and the press is ready for opening. The cakes or slabs of slime are usually 3 ft. square, and a 50-cake press 2 ins. thick contains 80 cubic feet, or about three tons. Messrs. Martin & Co. are now making presses 42 in. square for 3-inch cakes, the 50-cake press holding nearly six tons. When the presses are used in conjunction with the agitators, as many as from 10 to 12 charges may be put through in twenty-four hours, or, in other words, as much as 60 tons per day. According to Mr. J. Moss, the cost for filter pressing without any grinding power is 10s. 10·2d. per ton, the average residues being 1·95 dwt. per ton. The total cost of treat-

ment from the mine to the tailings dump, allowing for first cost in labour and supplies and second cost in repairs and renewals, is put down at 35s. 9'4d. per ton.

The Dehne machines can, if necessary, be constructed of gun-metal or wood, with all the channels and valves lined with material suitable for withstanding the corrosive effects of any particular material.

Details of the Diehl and Riecken slime processes, both of which are in operation at Kalgoorlie, will be found in Chapter XIII.

Roasting previous to Cyaniding.—With pyritic concentrates and even high grade pyritic ores, a higher percentage of the gold contents can always be obtained by roasting previous to cyaniding, and in the Cripple Creek camp of Colorado and Kalgoorlie in Western Australia this process is largely adopted for the treatment of sulpho-telluride ores.

By roasting, the cyanide solutions are kept freer from soluble salts than when treating raw ore ; a higher percentage of gold can be extracted at a smaller cost ; the time of treatment is shorter ; and it is found that in the case of clayey ores, roasting causes dehydration, thereby rendering them porous, and making filtration comparatively easy.

The author found that Moanataiari concentrates, which yielded only 30 per cent. of their values in the raw state, yielded, when roasted, 90 per cent., with a smaller consumption of cyanide.

In order to ensure success it is essential that the sulphides should be subjected to a "dead" roast.

A quick and satisfactory test to determine if the ore is dead roasted, and amenable to cyanide treatment, is described by Wallace Macgregor as follows* :—

"Take samples of the roasted ore at discharge end of furnace, cool, take from 100 to 250 grams, place in a beaker with 200 c.c. of water ; stir this by shaking for about a minute, then filter into a beaker or flask, and to the filtrate in the beaker add a small quantity of cyanide solution made up to the same strength as that used in the regular work of the plant. It is best to add the cyanide solution slowly and carefully, noting the result. If no cloudiness at all appears the ore is dead roasted, or at least well fitted for treatment by cyanide solution, and the consumption of cyanide will be normal.

"If a brown coloration takes place there are still some soluble salts of iron left in the ore, which will cause a somewhat higher consumption of cyanide, and may lead to a precipitation of compounds of ferro-cyanide in the zinc boxes. If, on addition of

* *Engineering and Mining Journal, and Mineral Industry, 1898.*

cyanide to the filtrate, a blue coloration, soon becoming a greenish blue precipitate, is formed, then the ore is very badly roasted, and one may look for a high consumption of cyanide, and the circulating solution will be made foul."

As to the economy of roasting ores, that must be determined for each individual case. There are oxidized ores and tailings from stamp mills which pay a fair profit by direct treatment with cyanide. These ores and tailings may be so low-grade that roasting would be out of the question, although if used it would increase the extraction of the gold on subsequent treatment with cyanide by probably 10 per cent. or more. There are cases of heavy sulphuretted ores and concentrates where roasting interferes with the extraction of the gold, and causes a very heavy consumption of cyanide.

The advisability of roasting ores previous to cyaniding should be carefully determined in each case, both by laboratory experiments and small plant tests, where 1 to 10 tons of dead roasted ore could be worked.

The main features of the cyanide process used for the treatment of the sulpho-telluride ores consist in drying the ore; dry-crushing in stamps, Griffin or other mills; roasting the pulverized ore in furnaces; amalgamation in pans; cyaniding sands, if any, in vats by percolation and slimes in agitators; filter-press separation of solutions from slimes, and final washing of slimes.

A detailed description of the plant and process used at some of the leading Kalgoorlie mines will be found in Chapter IX.

In the Ohinemuri Goldfields of New Zealand, the gold occurs in an extremely fine state, and hitherto dry-crushing and cyaniding have been used, the objection to wet-crushing being the difficulty experienced in treating the slimes which were always high grade.

The successful experiments conducted at the Crown mines with wet-crushing last year, and the subsequent adoption of this process at the Company's mill, have shown that, with careful manipulation, the simultaneous treatment of the slimes can be carried on with very little extra cost.

The following description of wet-crushing with cyanide solution has a special interest.* The ore is delivered to the rock-breaker without drying. With the ore there is fed into the mortars a constant stream of a stock cyanide solution of about .1 per cent. strength, regulated by stop-cocks near the boxes. The rate of flow is kept between 35 cwt. and 2 tons per hour, being gauged at the supply-tanks. As the stamps, which weigh 1000 lbs. with

* Thorpe, *Auct. Min. Standard*, Jan. 19, 1899.

96 drops to the minute, crush about 2 tons per stamp per day, the amount of solution is consequently about equal to the ore crushed. Previous experiments showed that a larger proportion of solution, while it increased the output, caused the pulp as well as the gold to be washed through the screens in a state too coarse for cyanidation. From similar considerations the discharge is raised to a height of 8 or 9 inches, the screens being 40-mesh. From the front of the boxes the pulp flows to a wheel-and-bucket elevator, and thence to the vat-shed. The vats are of the size usual in dry-crushing on those goldfields, viz., $22\frac{1}{2}$ ft. in diameter and about 4 ft. in depth, not including the space of 4 inches below the filter-cloth. The pulp flows into the centre of a wooden, two-armed distributor, driven by overhead gearing. Suspended to the arms are five sets of rectangular pieces of sacking, each set being composed of three or four pieces of varying lengths, so that they may be lifted as the vat fills. Their object is to keep the slimes on the surface of the sands from settling. These slimes are being continuously decanted off by an adjustable pipe into a receiving vat on the same level, provided with an agitator similar to that used in an ordinary settler. Thence they are pumped from time to time by a small centrifugal pump into the settling vats, also provided with light agitators. These vats are similar to the sand vats, but have the filter-cloth and frame removed. From these the clear solution is kept continually draining into a sump, whence it is pumped to the head of the precipitating boxes. After passing through the boxes it is pumped into the supply tanks, whence it feeds the stamps or is used for washes.

The sands, which contain a certain amount of slimes, are treated as ordinary tailings, percolating, with a vacuum of 20 inches, at the rate of about 5 to 8 tons per day. The extraction obtained is over 80 per cent., and with a few modifications in the treatment will probably be better still. The strength of the strong solution is .5 per cent., and the solution draining off is run through the towers with that coming from the slimes. In this way the stock solution is kept up to about .1 per cent. in strength, with little variation. The slimes give no difficulty in the solution of gold, the .1 per cent. solution being apparently quite strong enough. Each slime vat is filled with slimes till the latter settle down to about the middle of the vat, the proportion of solid matter in this state to the solution being about 13 : 20. The agitator is then started, and a weak wash added, equal, if possible, to the amount of slimes. In half an hour these are sufficiently mixed, the agitator is stopped, and the clear solution is gradually decanted off by adjustable, fan-mouthed pipes. In this way, neglecting, for the sake of simplicity, the amount of solid matter

in the slimes, the original amount of gold solution in the vat is reduced to a half. A second wash reduces it to a quarter, a third to an eighth, and so on. In this particular instance about 20 tons of slimes are given four washes, or from 15 to 20 tons of cyanide solution each, and one of water. The extraction of gold usually amounts to 90 per cent., a few exceptions being probably due to the imperfect decanting of the slimes from the sand vat, from which some of the coarser material might easily descend into the slime vat.

The method of working is as follows:—The ten-head employed in this process fills one vat with sand in three days. There being three sand vats, and the stamps being “hung up” during Sunday, seven days are available for treating the sand in each vat. For treating the slimes there are four vats. This allows the slimes from a day and a half’s crushing to be put into one vat, each of the four vats being filled in the six days. This gives five and a half days for washing, etc. The washes decant off, with the aid of lime, at the rate of from 15 to 20 tons per day. The extraction of gold in the precipitating boxes is so far very good. The chief advantages of this method over dry-crushing are: (1) Elimination of the expense of drying the ore; (2) increase in stamp duty; (3) economy in consumption of cyanide; (4) saving of labour in filling; (5) absence of dust. In the mill under review the stamp duty has been increased from one and a half tons to two tons per day per stamp; the consumption of cyanide decreased by about half. The “cyanides” formed in roasted ore are absent, while the strength of the solution used in the boxes is too weak to suffer much loss by agitation. Crude as are some of the appliances used in this process, the results justify the expectation that dry-crushing with cyanidation will soon be entirely displaced by wet-crushing with cyanide.

Sun-drying Slime Process.—At the Try Fluke mine at Kuaotunu in New Zealand, the slimes were dried in the sun, broken up, and then mixed with sands in the proportion of one part of slimes to two parts of sand. This method gives very satisfactory results, but can only be carried on in a very dry climate.

This slime process is in use in India, and in the United States at the Dexter plant, at Tuscarora in Nevada, where one part of dry slime is mixed with one part of sand and leached with a 0·20% solution of cyanide for three to five days, the solution running in at the top as fast as it runs out at the bottom until an adequate extraction has been obtained.

CHAPTER VIII.

TREATMENT OF CONCENTRATES.

With low-grade ores, it is generally more advantageous to omit close concentration and rather to classify prior to cyanide treatment. In all cases the best results are obtained from the cyanide treatment of concentrates when the pyrites and sand exist in the proportion of five to one.

Pyritic concentrates may be leached *by agitation* with cyanide, or simply *by percolation*, as with ordinary tailings. Both methods give satisfactory results.

LEACHING BY PERCOLATION.—At the Watersrand Goldfields the treatment of concentrates by cyanide has been largely adopted in preference to chlorination.

From the storage vats the concentrated material is taken to the leaching tanks, and subjected to the action of strong cyanide solution, containing from 0·4 per cent. to 0·6 per cent. of cyanide, for periods varying from twelve to eighteen days. In practice, the solution is allowed to slowly percolate through the concentrates, and it is then passed through the zinc precipitation boxes. It is again made up to the original strength, and allowed to percolate as before. This operation is continued until a satisfactory extraction is obtained. At the Crown Reef Cyanide Works the cost of this method of treatment is said not to exceed 17s. per ton of 2000 lbs.

Mr. C. M. P. Wright gives the following useful details of the cyaniding of concentrates by percolation at the Choukpatat gold mines.*

“The concentrates consist of 30 to 40 per cent. sulphides and 60 to 70 per cent. coarse sands: of the sulphurets, other than iron pyrites, 5 per cent. consists of franklinite, galena, chalcopryite, and a very little altaite. Of these minerals the franklin-

* Wright, *Inst. Min. and Met.*, London, Nov. 20, 1902.

ite is by far the richest, assaying from 7 oz. per ton and upwards; the galena holds practically no gold, and the chalcopyrite and iron pyrites vary from about 18 dwt. per ton to 2 oz. per ton, depending on the general value of the ore.

"Our mode of treatment is simple. After an alkaline or plain water wash, follows a weak solution wash (catch-sump strength 0.10 per cent. to 0.12 per cent. KCN) and then nine washes of 0.3 per cent. strength: the contents of vat are carefully turned over, and 0.3 per cent. solutions follow until for two successive days the effluent comes down to 0.26 per cent., when the treatment is considered complete. Two more solutions, catch strength of strong solution sump, usually 0.25 per cent., are followed by two weak solution 0.07 per cent. washes, and by a final water wash, or two if required.

"Every solution and wash that passes through the percolation vat passes into the same sump through the same zinc box. A careful check is kept of cyanide used, of solutions taken from weak solution sump, and of strong solution thrown on to tailings vat so that the level of the special sump is kept constant and at same time the somewhat loaded strong solution is exchanged for a cleaner weak solution.

"The check upon consumption of cyanide and zinc is complete and automatic. Usual tonnage charged monthly, 17 to 20 tons; duration of treatment, 24 days; extraction, 84 per cent. Formerly the zinc in zinc box was made up mid-monthly; this was found to be not necessary and wasteful of zinc. This zinc box is now carefully packed at the commencement of the treatment and left absolutely untouched till the clean-up: it is an eight compartment 1,000-ton per month box, and each compartment is filled."

Mr. Wright states that owing to the presence of copper in the concentrates, all the zinc became coated with that metal immediately after being put into use. The cost per ton of concentrates treated was 12s. 10d., and the extraction 84 per cent.

Leaching by Agitation.—At the Woodstock mine in New Zealand, the concentrates are treated by agitation in small wooden vats provided with mechanical stirrers fixed on a vertical steel shaft which is actuated by bevel-gearing, which, in turn, derives its motion from an intermediate shaft by means of a belt and suitable pulleys.

The concentrates have a value of £30 to £40 per ton, a large proportion of the value being in silver sulphide (argentite). They are agitated for thirty-six hours with a 4 per cent. solution of cyanide. Two pounds of lime are added for every ton of concentrates. The charge weighs $1\frac{1}{2}$ tons. The actual recovery is said

to vary from 90 to 94 per cent. at a cost of 18s. per ton for labour and material.

The leaching of concentrates by agitation was first introduced in New Zealand at the Sylvia mine by Dr. Scheidel in 1891, where the results were very satisfactory. Details of the plant and treatment will be found in the next chapter.

CHAPTER IX.

LEACHING BY AGITATION.

THE first attempt to introduce the cyanide process on a working scale, for the recovery of gold and silver from their ores, was made by the Cassel Gold Extracting Company, of Glasgow, at the New Zealand Crown mines, Karangahake, in 1889. The operations were under the supervision of Mr. J. McConnell. In the first plant, agitation formed a prominent feature, but in later years leaching by percolation became the more general and favourite method of treatment.

The new and extensive cyanide works of the Crown Mines Company, besides a percolation plant of twenty-four tanks, contained an agitation plant consisting of sixteen wooden tubs, or barrels, fitted with revolving paddles. The agitators were seldom used, preference being given to percolation.

At the concentration plant at the Sylvia mine, Thames Goldfield, an agitation plant was erected by Dr. A. Scheidel, in 1891, consisting of three agitators, 6 ft. in diameter and 6 ft. deep; three vacuum-filters, together with the necessary solution tanks, zinc extractors, and other appliances for cyanide treatment.

The ore was heavily charged with iron pyrites, and occasionally a small proportion of zinc-blende, argentiferous galena, and copper pyrites. It was wet-crushed in a 10-stamp Californian battery, classified in pyramidal boxes, and subsequently concentrated in jiggers, slime-tables, and buddles.

The concentrates of four grades were afterwards subjected to cyanide treatment by agitation. The extraction from the best slimes is said to have amounted to 96·45 per cent. of the gold, and 94·59 per cent. of the silver. The average extraction from all classes of concentrates amounted to 82·67 per cent. of the assay value.*

The strength of the cyanide solutions varied from 0·5 per cent. to 1 per cent., and the time of agitation from six to twenty-four

* *The Cyanide Process*, by Dr. A. Scheidel, p. 79.

hours. About 300 tons of concentrates were treated, and yielded over £10,000 value of bullion, but the cost per ton is not given.

The extraction is said to have been satisfactory until an excess of copper ore appeared in the concentrates, which rendered them unsuitable for cyanide treatment.

The cyanide plant at the Thames School of Mines, designed and erected by the author, is provided with an agitator which serves a double purpose, being used also as a dissolving vat when ores are being treated by percolation.

The agitator is similar to those used at the Crown and Sylvia works, consisting of an upright tub or barrel provided with a central revolving spindle set in a foot-step at the bottom. At the bottom end of the spindle is fixed a screw, consisting of four paddles or blades. The foot-step, being in the agitator, is subject to great wear and tear, and this forms a most objectionable feature, as it must be continually renewed. This difficulty is easily got over by fixing a hollow cone in the centre of the agitator and placing the spindle in this, the motion being applied from below, as in most grinding and amalgamating pans. An agitator of this kind was erected in 1894 by Dr. Scheidel at the Utica mine, Calaveras County, in California.*

The treatment of concentrates by agitation has already been described at the end of the preceding heading.

The Actual Extraction by Agitation.—The practice of the author was to charge the agitator with the cyanide solution made up to the required strength, using from 40 per cent. to 60 per cent. of the weight of the ore. The agitator was then set going, about fifty revolutions per minute, and the ore or tailings gradually fed in, until the charge was complete. The agitation was continued until a satisfactory extraction had been effected, which generally took from six to ten hours. Samples for assay were obtained from the agitator by means of a small tin at the end of a stout string. When the extraction was considered adequate, the agitator being still in motion, the stop-cock was opened and the pulp allowed to discharge into a percolation vat, where the solution was drawn off and the residues washed, aided by an artificial vacuum. The cyanide solutions were then passed through the zinc precipitation boxes.

Leaching by agitation possesses many advantages over percolation for the treatment of pyritic concentrates or rich tailings. The dissolution of the gold is much more expeditious, taking hours where percolation requires days; and with suitable material the extraction is always high.

* *The Cyanide Process*, by Dr. A. Scheidel, 1894, p. 89.

On the other hand, agitation requires motive-power, and from the nature of the process the charges must be small, in no cases exceeding a few tons. There is a prevalent belief that agitation causes an excessive consumption of cyanide by decomposition by atmospheric carbonic acid gas; but the author thinks this source of loss much exaggerated, and is certainly much less than it was in the early attempts at cyanide treatment when agitation was prolonged from thirty-six to forty-eight hours continuously.

The author has found, by many trials, that from six to eight hours' agitation is sufficient to effect the dissolution of the gold in the most refractory ores when reduced to a sufficient degree of fineness.

Numerous experiments during the progress of agitation have shown that the greater portion of the gold was dissolved during the first hour.

The rate of the extraction at the different hour-periods, during the treatment of the Monowai sulphide ore, is given below, and will be found instructive:—

Time of Agitation.	Gold Extraction %.
After 1 hour	85·4
„ 2 „	88·2
„ 3 „	90·6
„ 4 „	92·3
„ 5 „	92·3
„ 6 „	92·3

From the above it will be seen that the maximum extraction was obtained in four hours. This was a complex sulphide ore, containing sulphides of copper, zinc, iron, and lead. An analysis of the more mineralized portion by Mr. F. B. Allen, M.A., B.Sc., Director of the Western Australia School of Mines, gave the following results:—

Insoluble gangue,	90·15
Copper pyrites, .	3·78
Iron pyrites, .	4·40
Galena, .	0·25
Zinc-blende, .	0·26
Alumina, .	0·13
Water and loss,	1·03
	100·00

The bullion contents of this ore were, gold 1 oz. 5 dwts., and silver 14 oz. per ton. A 0·6 per cent. solution of cyanide was used for leaching by agitation, and the consumption amounted to 0·45 per cent., with an extraction of 92 per cent. A very large amount of copper was dissolved, and, becoming deposited on the zinc, caused much trouble in the precipitation of the bullion. The actual extraction was below 70 per cent.

The following interesting and instructive experiments by agitation with 0·25 per cent. and 0·33 per cent. cyanide solutions on "Martha" ore were kindly supplied by Mr. E. G. Banks, the metallurgist for the Waihi Gold Mining Company at Waihi:—

EXPERIMENTS on "Martha" ore, showing rate at which the precious metals were extracted by a 0·25 per cent. solution of KCy, and amount of KCy consumed.

		Gold.		Silver.		Saved per ct.		KCy
		oz.	dwt. gr.	oz.	dwt. gr.	Gold.	Silver.	used.
ORE BEFORE TREATMENT,		0	16 8	3	4 13			Per ct.
1.	,, after 2 hours' agitation,	0	9 19	2	15 20	40·1	13·5	·08
2.	,, " 4 "	0	8 4	2	13 22	50·0	16·5	·09
3.	,, " 6 "	0	6 14	2	7 10	59·7	26·5	·094
4.	,, " 8 "	0	4 14	2	4 40	71·4	31·2	·103
5.	,, " 10 "	0	3 12	1	19 20	78·6	38·3	·117
6.	,, " 12 "	0	3 12	2	3 1	78·6	33·9	·132
7.	,, " 14 "	0	3 6	2	2 11	80·1	34·2	·126
8.	,, " 16 "	0	3 6	2	2 11	80·1	34·2	·13
9.	,, " 18 "	0	3 10	2	3 3	79·1	33·2	·13
10.	,, " 20 "	0	3 1	2	4 8	81·4	31·3	·13
11.	,, " 22 "	0	2 15	2	2 8	83·9	34·4	·155
12.	,, " 24 "	0	2 11	2	2 11	84·9	34·2	·135

FINENESS OF ORE TREATED, 1·5% remained on a 40-mesh screen.
 14·5% " 60 "
 26% " 80 "

CHAPTER X.

ZINC PRECIPITATION AND TREATMENT OF GOLD SLIMES.

THE zinc for bullion precipitation is used in thread-like turnings, as this form gives the most surface for the least weight. It should be free from arsenic or antimony, although a little lead is an advantage, as it causes more rapid precipitation by forming a voltaic pair with the zinc.

As a general rule, one cubic foot of zinc turnings will precipitate the gold from two tons of solution in twenty-four hours.

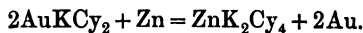
Zinc on which a film of bullion has been precipitated is more active than pure zinc, and it is therefore advisable to replace the zinc dissolved in the upper compartments by moving the zinc forward from the lower compartments, and adding fresh zinc to the latter.

In practice, the cyanide solution is allowed to slowly drain through the zinc in the precipitating boxes. The rate of flow is soon determined by actual experience. It is generally found that 85 per cent. to 95 per cent. of the bullion will be precipitated in the first three boxes.

The solution, after leaving the boxes, should not contain more than six or eight grains of gold to the ton.

Zinc, in the form of zinc-dust and zinc-fume, is in use for precipitation, particularly the former, but neither seems likely to supersede zinc turnings, which possess the important advantage that they afford a continuous method of precipitation, whereas dust and fume have to be applied in charges in solutions collected in separate vats. Furthermore, the zinc extractor-boxes, when once in good working order, require very little attention except at the periodical "clean-up."

The principle of precipitation of gold by metallic zinc is based on the fact that cyanide has a stronger affinity for zinc than for gold, as shown by the following equation:—



By the above reaction it will be seen that 1 oz. of zinc should precipitate 6 oz. of gold, but in practice it is found that from 4 oz. to 12 oz. of zinc are required to precipitate 1 oz. of gold. The reactions which take place in the zinc precipitating boxes are at times most varied and perplexing, especially during the treatment of pyritic tailings or acid mineralised ores.

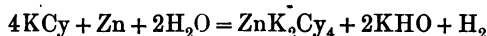
Part of the excessive consumption of zinc is no doubt due to decomposition by free cyanide, as may be ascertained by testing the solution for available cyanide before entering and after leaving the zinc precipitating boxes; but the consumption and consequent loss of cyanide by this cause is much less than generally supposed, and in all cases insufficient to account for the great waste of zinc.

Zinc is soluble in an aqueous solution of potassium cyanide without evolution of hydrogen. In the extractors, the cyanide comes in contact with so extensive a surface of zinc that a large quantity of that metal must pass into solution, but curiously enough the consumption of free cyanide in the extractor does not correspond with the consumption of zinc, and we can only conclude that a process of regeneration takes place in the extractor. It is quite certain that the fouling of cyanide solutions with that troublesome inert alkaline substance, zinc-potassium-cyanide, takes place in the passage of the solutions through the extractors.

It might naturally be expected that zinc would accumulate in the cyanide solutions to a detrimental extent, but this is found in practice not to be the case. The zinc does not accumulate to any extent, a result in all probability due to the action of the sulphides contained in the ore and cyanide which cause its precipitation as a sulphide of zinc.

The precipitation of the gold, doubtless from electro-chemical causes, is always more rapid and complete from moderately strong than from very weak cyanide solutions, but under all circumstances the solutions must be distinctly alkaline to ensure a satisfactory precipitation.

It has been suggested by some chemists that this is due to nascent hydrogen, liberated by the action of the free KCy on the zinc, taking the place of the gold, according to the following equations:—

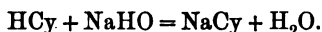


and



The liberated hydrocyanic acid is capable of combining with any free alkali present, and thus there would be no loss of the

cyanogen combined with the gold. This reaction is shown by the following equation :—



Hydrogen gas is always evolved when gold is precipitated, and the gentle action of the gas bubbles, as they rise to the surface in the zinc boxes, is an indication of satisfactory precipitation.

During the treatment of pyritic tailings at Kuaotunu, the unsatisfactory precipitation of the gold was for some time a source of much trouble to the chemists in charge of the cyanide works, but this difficulty was overcome by making up the strength of the solution before entering the extractor to something like the original working strength.

In practice, this was effected without any extra trouble by simply placing a barrel containing a strong solution of cyanide at the head of the extractor, and allowing a steady drip into the cyanide solution, in the top compartment, which was filled with a filter of sand and gravel. By testing the cyanide solution a few times, the rate of drip to bring it up to the required strength was easily determined.

The author used this method with success in the treatment of cupriferous ores from the Monowai mine in the Hauraki Goldfields, in 1895.

It was found that the dissolved copper was precipitated much more rapidly from a weak solution of cyanide than from a strong one. In order to overcome this, the solutions were made up to the original working strength. This method has now been superseded by the lead acetate pickling process.

In the treatment of slimes by decantation and in filter-press processes, but especially in the former, there are formed large volumes of excessively weak cyanide solutions containing gold. It was a matter of early cyanide experience that zinc precipitation from such weak solutions was very imperfect, in fact far inferior to electrical precipitation.

The discovery was soon made that the lead-couple used for the precipitation of gold from solutions containing copper was also very effective for the precipitation of gold from extremely dilute cyanide solutions, and the practice in South Africa and elsewhere in slimes plants is to pickle the zinc, before use, in a trough containing a 10 per cent. solution of lead acetate, until all the zinc is covered with a black coating. The precipitation effected by this lead-couple is almost perfect, only a trace of gold, as a rule, finding its way into the sump.

At the Camp Bird mines, as described in the following page, a

zinc-mercury couple is used with very satisfactory results for solutions containing copper, and is stated to give better results than the lead-couple.

It is the practice at some plants to allow a dilute solution of lead acetate to drip slowly into the head of the zinc extractors, but the practice is not to be commended, on account of the difficulty of regulating the rate of flow. Besides, by this method an excessive amount of lead is certain to find its way into the bullion.

It is now the practice when the cyanide contains sulphides to sprinkle lead acetate on the top of the tank. By this means the alkaline sulphide is decomposed, and the cyanide freed from sulphide by the precipitation of lead sulphide.

When copper is present in the solution, it soon covers the zinc with a bright metallic coating, which begins in the lower boxes, and gradually encroaches on the upper ones. When the zinc is coated with copper, the precipitation of the gold is very slow and imperfect. By increasing the strength of the solution to near the working strength, before it enters the boxes, the copper may be largely kept in solution.

When the ore or tailings contain copper, the supplies of fresh zinc should only be added when the strong solution is passing through the extractor. By attending to this much of the copper can be kept in solution, with a correspondingly satisfactory precipitation of the gold.

It should, however, not be forgotten that when there is much copper present in the ore, it becomes imperative to allow the precipitation of the copper with the gold with the object of keeping the copper contents of the solutions constant. In such a case to keep the copper in the solutions would soon result in their becoming overcharged with copper and thus useless for gold extraction purposes.

If the zinc turnings be placed in a solution of lead acetate, of, say, 10 per cent. strength, they will become covered with a porous coating of lead. This lead-coated zinc, by its electro-chemical energy, will effect the perfect precipitation of the gold, and leave the copper, even from the weakest solutions, unprecipitated. The resulting bullion obtained by this means is, however, always highly charged with lead.

At the Camp Bird mills, Ouray in Colorado, the waste solutions are very weak, of poor value, and contain some copper. The lead acetate method was tried without much success and abandoned in favour of a mercury-couple which is obtained by immersing the zinc in a weak solution of mercuric cyanide until it is coated with mercury. The mercury coated zinc is stated to give a very fair

precipitation, while the mercury is recovered and does not pass into the bullion as is the case with lead.

Experience has shown that ores containing much copper are not adapted for cyanide treatment, firstly, on account of the undue consumption of the cyanide; and, secondly, on account of the difficulty of precipitating the gold in the presence of the base metal; moreover, by continued use, the stock and sump solutions become charged with copper, and thus rendered useless for all practical purposes, such as washing, or forming the basis of working solutions.

Occasionally an inert gritty, greyish-white, porous precipitate of zinc cyanide forms on the zinc in the precipitating boxes. The reactions which lead to its formation have not yet been satisfactorily explained, but, whatever they may be, its presence is always accompanied by loss of cyanide and imperfect precipitation of the gold. This precipitate is seldom seen excepting in the treatment of pyritic ores and tailings. It can generally be prevented by a careful preliminary washing, and treatment with lime instead of caustic soda.

On the other hand, when a too free use of lime is made to reduce acidity, an incrustation of lime will form on the zinc and thus prevent satisfactory precipitation.

In some cases there may be, in the presence of organic compounds, an excessive and injurious evolution of hydrogen. During the treatment of decomposing pyritic tailings at the Great Mercury Cyanide Works, Kuaotunu, the evolution of hydrogen gas was so vigorous that it lifted the zinc out of the precipitation boxes, forming a thick froth. On this occasion the precipitation of the bullion was very imperfect and unsatisfactory, and suggested polarization.

When a scum forms on the surface of the solution in the precipitation boxes, both it and its cause should be removed without delay. In the case of accumulated tailings it will generally be found to be caused by the presence of decomposing organic matter, and the application of an oxidizing agent often exerts a beneficial effect.

In practice the zinc shavings are first placed in the weak solution box, and afterwards transferred to the medium, and thence to the strong. In the weak and medium solution precipitating boxes, the gold becomes plated on the zinc, and less zinc is destroyed than in the strong. The solution from the extractor boxes, containing only traces of gold, is returned to the solution tanks, where, if found necessary, its strength is made up by the addition of cyanide.

The precipitation of the gold from very weak solutions, especi-

ally in the presence of copper, has always been a difficulty with zinc, but of late years success has, to a great extent, been obtained by pickling the zinc, before use, in a solution of acetate of lead, and adding fresh cyanide at the head of the box, as was done by the author in 1895.

This method of precipitation has been in use in the Lydenburg district for over five years, and is now largely in use at the Rand, to assist the precipitation of the gold contained in the extremely weak cyanide solutions, which necessarily accumulate in large volumes in the treatment of slimes.

The following interesting particulars of the operations at the latter place have been given by Mr. T. L. Carter* :—

A very important point is the preparation of the zinc. After it has been cut on the lathe it is taken to a trough which contains a solution of acetate of lead of about 10 per cent. strength. The zinc is thoroughly washed and stirred in the solution until it becomes of a dark hue. If it is not thoroughly stirred only the outside of the mass of zinc will become coated, the inside remaining quite bright. An empirical way to ascertain whether or no the solution is strong enough is to look at the zinc, and see if it is sufficiently and thoroughly coated. After thus preparing the zinc it should be placed in the box, and covered with the auriferous solution as quickly as possible, since leaving it in the air seems to affect it adversely.

The next important point is the addition of free cyanide at the head of the box. Twenty pounds of KCy are dissolved in an iron tank, holding about 75 gallons water. This $2\frac{1}{2}$ per cent. solution is allowed to run freely into the auriferous solution entering the box for a period of about four hours, raising the strength of the solution passing through from .007 per cent. to .025 per cent. When the 20 lbs. of KCy are finished, another 10 lbs. are dissolved as before, and freely run into the box, taking a period of six hours, and raising the solution about .007 per cent. higher. By no means should the addition of the cyanide as described be neglected, for it is found absolutely necessary at the commencement to let this free cyanide run into the box. Twelve or fourteen hours after starting a slow drip is allowed to fall into the solution as it enters the box, bringing up the strength of the solution going through the box from .007 per cent. to .008 per cent., and this is dropped in through the run of the box. The precipitation is almost perfect. On account of the lead present in the slimes, the smelting gave much trouble at first, but, after many experiments, the following flux was found to give the best results :—

* *Jour. Chem. and Met. Soc. S.A.*, No. 9.

Borax, ...	60 per cent.
Nitre, ...	19 "
Sand, ...	11.5 "
Soda, ...	7 "

The precipitation of gold by zinc, results in the formation of a double cyanide of zinc and potassium, and the continual use of the same solutions would lead to the belief that the working solutions would in time become charged with zinc salts. In actual practice it is found that this is not the case to any great extent.

Feldtmann states that, under favourable conditions, the zinc-potassic cyanide is of itself capable of dissolving gold from its ores. He considers that the small quantities of alkaline sulphides present in commercial cyanide, or formed by the action of cyanide on metallic sulphides, serve to precipitate a portion of the zinc as the insoluble sulphide.

In cases where ores contain considerable proportions of metallic sulphides, soluble in cyanide solution, sufficient alkaline sulphide may be formed to precipitate a portion of the dissolved gold.

To prevent any loss in this direction, Mr. J. S. McArthur suggested the addition of a solution of some soluble lead or other metallic salt which is known to form an insoluble sulphide in alkaline solutions, and this practice is now general when the cyanide contains sulphides. In this case, however, it would be advisable to avoid an excess of the lead, or other salt, so as to prevent possible complications in the extractor. The exact amount of salt required can be determined in the laboratory.

The Clean-up.—The periodical clean-up takes place once or twice a month. The first operation is to pass a current of clean water through the zinc boxes, so as to remove the cyanide solution, which is injurious to the workmen, often causing their arms to become covered with painful red boils.

The trays holding the zinc are then moved up and down in their compartments so as to allow the fine gold precipitates, and fine particles of zinc, to fall through the sieve and settle in the bottom of the box. The contents of the trays are then placed in a large trough, provided with an easily removable false bottom of finely perforated iron. The zinc is gently teased out and rubbed in this trough, which is partly filled with clean water, and in this manner as much as possible of the adhering gold is removed. After all the gold has settled, as a slimy mass, the water is syphoned off. The gold slimes remaining in the extractor are sluiced through plugholes into the side launder, or into the bottom compartment and collected in a trough. The fine

slimes or precipitates are rapidly settled by the addition of a little powdered alum to the solution.

In large cyanide works the precipitates are dried in a small filter-press or vacuum-filter. The discoloured zinc shavings are now returned to the precipitation boxes, fresh zinc being put in the lower compartments. The gold still remaining on the zinc is recovered at the next clean-up.

The gold slimes are treated in one of three ways, namely :—

1. By **smelting** with suitable fluxes.
2. By **sulphuric acid** method.
3. By **lead smelting** method.

The smelting method is the oldest and still has many adherents. It is, however, laborious, slow, and attended with loss of bullion. It is gradually being superseded by the sulphuric acid method, which in its turn seems likely to be superseded by the lead smelting process.

1. **Smelting Process** :—*Roasting the Precipitates*.—The dry precipitates are roasted at a low heat, with free access of air. The object of the roasting is to oxidize the zinc in the slimes, and thus cause it to combine with the fluxes in the subsequent smelting, and thereby leave the bullion as fine as possible.

In Australia and New Zealand cyanide works the roasting furnace often consists of a large flat cast-iron plate, with raised edges. It is built over a small grate or furnace, and a hood of light sheet-iron is placed over the roasting place, so as to carry off the zinc fumes.

The roasting should be conducted at a moderate heat, *i.e.*, it must never rise above a *dull red*, and the precipitates must be stirred continuously so as to expose fresh surfaces to the action of the atmospheric oxygen. During the early part of the roasting, dense white vapours of zinc oxide are given off, but as the operation advances they are observed to diminish and finally to cease entirely when the reaction is complete. Time, from one to two hours.

Mr. Feldtmann has found that the oxidation of the zinc is facilitated by the addition of a little nitre, say from 3 per cent. to 10 per cent. He suggests that it should be applied to the slimes as a strong solution before the drying, so as to get thoroughly mixed with the whole mass. The nitre not only helps to oxidize the zinc, but is also said to assist the subsequent fluxing by uniting with the zinc oxide, and forming a zincate of potash, which is not so readily reduced by the plumbago of the crucible as the oxide. At many works the nitre is added to the dried slimes in a powdered form ; of course less nitre must be

used than is necessary to oxidize all the base metal present, as any free nitre remaining would rapidly destroy the plumbago crucible during the smelting process. Besides rendering the bullion fine, the nitre roasting gives a cleaner slag and greatly hastens the fusion.

When stirring and removing the roasted precipitates, care must be exercised to avoid a loss of bullion in the form of dust.

Smelting the Oxidized Precipitates.—The roasted precipitates are now placed on a large, shallow iron tray, mixed with the necessary fluxes, and fused in plumbago crucibles. The following fusing mixtures have always given satisfactory results:—

	Clean Precipitates Little Zinc.	Much Zinc and Little Sand.	Very Sandy.
Precipitates, . . .	100	100	100
Bicarbonate of Soda, .	6	20	50
Borax, . . .	50	50	30
Sand, . . .	3	15	—
Fluor-spar, . . .	—	—	2

The chief essentials in a slag are fluidity, small bulk, and non-corrosion. The first is conferred by the “fused” borax and the last by the sand.

It is desirable that the fluxes used should be free from moisture, so as to avoid loss due to the escape of steam through the charge.

Smelting Acid-Treated Slimes.—Acid-treated gold slimes are necessarily of a basic character, hence an acid flux must be used so as to produce a neutral or non-corrosive slag. Manganese dioxide is now generally added for carrying base metals into the slag. Messrs. E. H. Johnson and W. A. Caldecott have shown that in a state of fusion it is even a more active oxidizer than nitre although it contains much less available oxygen.*

The following is the basis of flux used by these authors, the proportions being varied, within the limits, as varying conditions required:—

Slimes, ...	100 parts.
Fused borax, ...	20 to 35 „
Manganese dioxide, . .	20 „ 40 „
Sand, ...	15 „ 40 „

The addition of soda to an already basic material was considered unnecessary. When sulphates are present, a little fluor-spar is found to assist the fusion: a few preliminary trial fusions should be made to ascertain the best proportions of the fluxes.

* *Jour. Chem. and Met. Soc. S.A.*, July 1902.

The authors quoted above state that a matte or base-looking bullion indicates too little manganese dioxide, whilst too much manganese dioxide yields an infusible slag or one containing much silver. It has long been known that manganese dioxide in fusion carries off silver, and for this reason in smelting slimes containing commercial values in silver, it must be used with care, or not at all.

Plumbago crucibles with removable clay-liners are generally used for the fusion of these charges. Crosse uses silicate of soda as a source of silica and clay-lined crucibles instead of removable clay-liners.

In works where large quantities of slimes have to be smelted, Nos. 50, 60, or 70 plumbago crucibles will be required.

The Actual Fusion.—The crucible, previously annealed, is placed on a flat brick resting on the bars of the furnace. A priming of borax is then placed in the crucible, and over this a charge of precipitates; fresh additions of precipitates are made as the charge fuses and subsides. When the crucible is two-thirds full, the slag is skimmed off and fresh portions of precipitates added until it is three-fourths full of molten bullion.

The crucible is now removed from the furnace, and the contents poured into ingot moulds which have been previously *well heated* and carefully oiled with the best olive oil. All excess of oil should be wiped out of the mould before pouring the metal.

The melting furnace may be constructed to hold two or three crucibles at the same time. It should be built of the best materials, as the heat required to melt the *slime mixture* is higher than that for ordinary smelting.

At the Langlaagte Cyanide Works the slimes, mixed with the fluxes, are charged into No. 50 plumbago crucibles, and melted in a reverberatory hearth furnace which holds 22 crucibles at the same time. The time required for melting varies from one and a half to three hours, according to the character of the materials and the temperature of the furnace.

The slag resulting from the smelting of slimes always contains a small proportion of gold. It is, therefore, generally pulverized in a single stamper, or in a small mill, and afterwards amalgamated with mercury. In some cases it is sent to the smelting works for treatment.

The ingots of bullion, obtained from the first smelting, are re-melted with borax; and, since gold forms but a very imperfect alloy with zinc, this second melting should be conducted at as low a temperature as possible so as to obtain an approximately uniform bar of bullion.

The zinc slimes generally contain from 30 per cent. to 65 per

cent. of bullion, the fineness of which, after melting, generally varies from 600 to 900.

The clips for assaying should be taken from different parts of the bar so as to obtain a representative sample for valuation, but the dip sample is always the most reliable.

McBride gives the cost of smelting gold slimes at 2d. per ounce of fine gold. At a clean-up giving 718 ounces of bullion, 817 fine in gold, the costs were as follows:—

Borax, ...	£1	5	0
Soda, ...	0	6	7
Fluor-spar, ...	0	5	6
Nitre, ...	0	2	6
Pots, ...	1	1	9
Coal, ...	1	16	0
Labour, ...	1	2	8

£5 19 8 = 2d. per oz. fine gold.

2. Refining by Sulphuric Acid.—This method is commonly used in cyanide works in America and South Africa, but has not yet been adopted to any extent in Australia or New Zealand. The acid treatment of the precipitates is a simple enough operation, and was occasionally used by the author for the refining of small parcels. The necessary apparatus consists of shallow wooden tubs, or vats.

The operation is conducted as follows:—Clean water is passed through the zinc extractor, for some time, to remove all traces of cyanide. The precipitates are then removed from the boxes and placed in the first vat, with a sufficient quantity of dilute sulphuric acid. The acid should not be too strong, nor yet too weak; a mixture consisting of ten parts of water and one part of strong acid answers well.

The quantity of dilute acid will depend on the proportion of zinc present in the precipitates. With 50 per cent. of zinc, about six parts of the acid mixture to one of the precipitates will be required; and with very zincy precipitates, from ten to twelve parts.

The mass in the tub should be stirred occasionally, and then allowed to settle. Heat is generated, and large quantities of hydrogen gas liberated by the action of the acid on the zinc.

When the undissolved precipitates have been allowed to settle, the clear liquid should be removed by decantation into a second tub, and thence finally, after an interval, into a third tub. By this means any fine particles of bullion which have escaped in the

first decantation will be secured in the second tub; and that which has escaped during the second decantation will be found as a fine sediment in the third tub.

The bullion should now be washed in the tubs with clean hot water, to remove all base soluble sulphates and any free acid remaining. Then remove the bullion slimes and dry on the vacuum-filter. When dry, subject to an oxidizing roasting on a shallow iron pan, for an hour or so, to oxidize any base sulphates present.

Next, mix with 5 or 10 per cent. of borax glass, according to the amount of zinc oxide still present, and fuse in a plumbago crucible in which a priming of borax has already been placed. As the charge fuses and subsides, fresh portions of bullion should be added, until the crucible is three parts full of melted metal. To permit of this being done the slag can be skimmed off from time to time.

If manganese dioxide is used in the flux it will be necessary to protect the crucible from corrosion by using a clay-liner as described in the preceding pages under the heading devoted to the "Smelting of acid-treated Slimes."

The bullion is generally from 850 to 900 fine, but with a little extra trouble can be worked up to 950.

With suitable appliances, this process possesses many advantages over the smelting process. It occupies less time, produces finer bullion, and, properly conducted, costs less.

When large quantities of precipitates have to be dealt with, the method of settling the slimes and decanting is too slow and expensive. In this case, the separation of the slimes from the acid solution, as well as the subsequent washings, must be effected in a vacuum-filter, or a filter-press, as practised in America and South Africa.

The filter used for the purpose is a wooden box, two or three feet square. It is provided with a filter webbing, or cloth, of fine canvas or twill duck, resting on a grating of wood, and fixed with slips of wood, so as to be easily detached for washing.

The false bottom, below the webbing, must be 15 in. or 20 in. deep, and provided with a solution gauge, the upper limb of which should be 2 in. below the air-exhaust pipe connected with the vacuum boiler.

Care must be taken to draw off the acid solution and washings by a plug-hole before they rise to the level of the air-exhaust pipe, which is placed immediately before the filter-frame.

When the acid solution is diluted to half its strength before filtering, the webbing lasts for several operations. A Johnston filter-press does the washing well and expeditiously.

Acid Treatment in South Africa.—The following description is an abstract of a paper read by E. H. Johnson before the Chemical and Metallurgical Society of South Africa. It describes the process employed at the Princess Works, where the slimes are submitted to acid treatment before smelting, and will be especially interesting for its figures of costs and metallurgical results:—

At the Princess Works the slimes from the zinc boxes are separated from the solution drawn off with them by the aid of a vacuum-filter. A water wash is passed through until the slimes are free from cyanide. The gross weight of the slimes, including moisture, is then taken, by weighing the buckets of moist slime during transference to a large sheet-iron tray placed alongside the acid tank, to determine the amount of sulphuric acid necessary to destroy the zinc.

Having found the approximate weight of slimes to be treated, sufficient water is run into the acid vat to form, on the addition of the acid, a 10 per cent. solution. One pound of acid for every pound of moist slime gives good results. This would be equivalent to about $1\frac{1}{2}$ lb. of acid to the pound of slimes, dry. The acid is then added, and the vat closed down tightly.

The stirring apparatus is kept continually moving during the time of feeding in the slimes, which are fed in gradually in the same condition in which they were taken from the filter-vat. It is beneficial to keep up a continual stirring for at least half an hour after the action has apparently ceased.

After all the slimes are in the acid, a jet of water is turned into the hopper to wash down any adherent slimes, and everything that has been used in the cleaning-up of the boxes, etc., is well washed in the same jet during removal. The vat is then filled with water and allowed to settle.

Working with dilute acid, and not heating, a perfect settlement takes place within an hour. When heating with a steam jet, settlement was much more difficult.

The washing is done by syphoning off the clear liquor, and filling the vat repeatedly with water, until the solution is neutral to litmus paper—usually four or five washings. It is well stirred at each refilling by means of a long wooden paddle, a rotary motion being given to the water. This causes the slimes to collect in the middle of the vat, and reduces the risk of loss during syphoning—the syphon being let down at the side. A sample of the washings taken continually during syphoning off showed, on careful assay of a large sample, 13 gr. of gold per ton of solution.

The drying of the resultant gold slime is conducted on an open drying hearth in large cast-iron enamelled dishes. The cakes are subsequently broken up and transferred to small sheet-iron trays

in thin layers, and subjected to an increased heat. When cool the slimes are ground, fluxed, and transferred to the crucible. It fuses quietly and with but little fume, and normally yields 50 to 60 per cent. of the weight of slime as bullion.

The average fineness of last year's bullion was, according to the Work's assays, 821·9 and 819·6, according to London returns. As a deduction of 2 milliemes is made on 800 bullion, this leaves an actual difference with London of 0·3 milliemes.

The slag, after panning out the prills (of which there is very little), assays 23 oz. per ton, and one ton of slag has been accumulated in two years for an output of 11,627 oz. of fine gold, which is equivalent to a little under 0·2 per cent. of the total gold produced.

The cost of reduction, including acid, is 6·7d. per fine ounce, made up as follows, taking an actual "clean-up" as a basis:—Dry weight of zinc-gold slimes, 504 lbs.; dry weight after acid treatment, 100 lbs.; 672 lbs. of acid at 4½d., £12, 12s. 10d.; 66 lbs. of borax at 37s. 6d. per cwt., £1, 2s. 1d.; 9 lbs. carbonate of soda at 2½d. per lb., 1s. 10d.; 9 lbs. fluor-spar at 4d. per lb., 3s.; 5 bags coke at 8s. 6d. per bag, £2, 2s. 6d.; 1 No. 60 crucible, £1, 7s. 6d.; total, £17, 9s. 9d.; yield, 620 oz. fine gold, or 6·7d. per fine oz.

From information gathered from reliable sources Mr. P. S. Tavener states that the cost of sulphuric acid refinement of slimes in South Africa is as follows* :—

- a. Sulphuric acid treatment from 4d. to 8d. per oz. of fine gold.
- b. Smelting cost exclusive of labour and furnace wear and tear 4d. to 6d.

Approximately this gives a total cost of 1s. per oz. of fine gold produced.

Acid Treatment in America.—I am indebted to Mr. Godfrey Doveton, one of my former assistants, for the following particulars of the acid treatment at the Camp Bird Mills, Ouray, in Colorado :—

"The gold slimes are refined by sulphuric acid in a wooden vat 12 ft. in diameter and 2 ft. deep, coated with paraffin paint. They are not subjected to heat, sufficient being generated on the addition of the acid to the moist slimes. About twelve hours are occupied in dissolving the zinc, and some forty-eight hours in washing out the soluble sulphates. The washing is much expedited by the use of hot water, at least for the first six washings. About fifteen wash waters are usually required. After the first two washes, having a high specific gravity, are removed, the time

* Tavener, *Jour. Chem. and Met. Soc. S.A.*, Oct. 1902.

allowed for settling is one hour. Usually it takes from eight to twelve hours to remove the first two washes, as the slimes do not settle so readily in the heavy acid liquid.

"All the washings are stored in a large settling-tank, which is cleaned up at long intervals. The settling-tank, after a year's run, yielded bullion worth £20.

"The first two or three wash waters on assay are found to yield from 10 to 20 cents per ton, but subsequent washes contain from 5 cents to a trace,—only apparent on assay of large evaporations.

"The gold slimes are partially dried by filter-press, and are then transferred to a hearth calcining furnace and calcined. The hearth of furnace has an area of 36 square feet. The furnace-doors are securely padlocked, and the furnace serves the double purpose of a safe and calciner.

"From the furnace the slimes are removed as required, and charged into graphite crucibles, and smelted, having been previously mixed with 50 per cent. of a flux composed of the following parts:—

Soda,	.	.	.	.	2 parts.
Sand,	.	.	.	.	1·5 "
Borax powder,	.	.	.	.	4 "
Sulphur,	.	.	.	.	as required.

Thus 200 lbs. of slimes require 100 lbs. of flux. The charge is smelted with frequent skimming in from ten to sixteen hours for each crucible.

"The slimes contain a considerable quantity of copper, but comparatively little zinc, consequently little or no nitre is used; but we use instead a certain quantity of flowers of sulphur, which readily converts the copper to a matte or regulus, leaving the bullion comparatively fine. The regulus, which by the way is usually from 15 to 25 per cent. of the weight of the bullion, was found to contain, on assay—

Gold,	.	.	.	23·4 oz. per ton.
Silver,	.	.	.	408·0 "
Copper,	.	.	.	60%
Zinc,	.	.	.	3%

"The slag resulting from the bullion melting is ruby red in colour, and contains—

Gold,	.	.	15 oz.	} Assayed without removing shots or prills of bullion.
Silver,	.	.	265 "	
Copper,	.	.	6 to 8%	
Zinc,	.	.	12 " 16%	

"The bullion averages about 800 to 850 fine, the impurity almost entirely copper with a little zinc.

"A shipment of 3000 lbs. of slag and regulus mixed was lately sent to the smelter and netted us £120."

3. Tavenor Lead-Smelting Method.—This process was first made use of by Mr. P. S. Tavenor in August 1899. Since August 1901 it has been in continuous operation at the Bonanza Mines, Limited, Johannesburg, and already the process has been adopted by many leading mines at Johannesburg. The adoption of the lead-smelting of gold slimes marks a notable advance in cyanide practice. The old smelting process is laborious, and always likely to entail serious losses of gold where large quantities of slimes have to be handled, while the sulphuric acid method is cumbersome, slow, and costly.

The obvious advantages of lead-smelting compared with the sulphuric acid process may be briefly summarized as follows :—

- a. Saving of cost per oz. of fine gold produced.
- b. No by-products.
- c. Less liability to loss in handling slimes.
- d. More gold actually produced from a given weight of slimes.

The essence of every metallurgical process is its cost, and judged by this principle, lead-smelting possesses a marked advantage. In South Africa, where the sulphuric acid method has, perhaps, its greatest application, the average cost, according to Tavenor, is not less than one shilling per oz. of fine gold produced. The lead-smelting process costs threepence per oz. of fine gold, including all charges, which means a saving of ninepence per oz. compared with sulphuric acid method.

Thus the lead method at threepence per oz., in a mine producing 2500 oz. fine gold per month from cyanide works, would effect a saving of costs amounting to £93, 15s. per month, equal to about £1000 a year.

The Tavenor lead-smelting process is cheap, rapid, and efficient compared with other processes, and with some modifications in practice is likely to be universally adopted.

The following working details of lead-smelting as practised at the Bonanza mine are extracted from a paper read by Mr. Tavenor before the Chemical and Metallurgical Society of South Africa last October.*

Mr. Tavenor says that he can best describe lead-smelting by comparing it to a scorification assay conducted on a large scale, for the reason that the zinc slimes are melted and the gold

* *Jour. Chem. and Met. Soc. S.A.*, Oct. 1902.

recovered in lead bullion. The lead bullion is then cupelled, or to use a better term, refined.

"The 'clean up,'" he continues, "is conducted in the ordinary way, with the exception that all the precipitate is at once pumped from the 'clean-up' tub into the filter-press. The fine zinc which remains at the bottom of the 'clean-up' tub is heaped up on one side and allowed to drain for about half an hour, and is then ready for the smelting room. The filter-press is cleaned out and the cakes taken in their moist condition to the furnace, and there both slimes and fine zinc are put in trays into a drying oven, and sufficient time is allowed to warm through, fifteen minutes for each tray being sufficient. Care is taken to keep the fine zinc separate from the filter-press slime, and on no account should they be allowed to get mixed.

"In charging the furnace, the slime is first dealt with. After warming in the drying oven, it is at once rubbed through a sieve, four holes to the linear inch, and then roughly weighed for fluxing, the necessary fluxes having been previously mixed.

"The slime is mixed with the fluxes and passed through a sieve to ensure thorough mixing. It is then shovelled direct into the furnace. When all the filter-press slime is fluxed and charged, the fine zinc is dealt with in the same way, and put into the furnace on the top of the slime in order to prevent loss by dusting, and also to have the greater portion of litharge present on the top of the charge. The fluxes used are residue assay slag and commercial litharge. The former costs nothing, for unless used in this manner it would be thrown away."

Discussing the question of fluxes, he says:—"I have found that the following, with little variation, will give satisfactory fusion and clean slag:—Slag, 25 to 30 per cent., made up of 10 per cent. assay slag; the balance, equal quantities of old slag and scalings from the pots of previous smelt. In the event of the lead-smelting method being adopted, I should like here to point out the advantage of storing the slag now being obtained from crucible smelting, since in the lead process its gold contents are converted into bullion free of cost. If it were not treated, clean slag would have to be used. During the last year I have been able to deal with several tons of this material assaying over fifty ounces fine per ton, left behind on the mine after the Boer occupation. The same applies to clay liners and anything else carrying gold, now termed by-products. With a lead-smelting furnace the word by-product is forgotten, since none is obtained."

The quantity of litharge to be used will depend on (a) value of slimes, (b) weight of charge in furnace, and (c) the percentage of gold required in the resultant lead bullion.

Mr. Tavener gives the following proportions of fluxes for gold slimes:—

Gold slimes,	. . .	100 parts by weight.
Litharge,	. . .	60 "
Assay slag,	. . .	10 to 15 "
Slag previously used,	. . .	10 to 15 "
Sand (SiO_2),	. . .	5 to 10 "
Sawdust,	. . .	1% of weight of litharge.

For fine zinc he uses the following proportions:—

Fine zinc,	. . .	100 parts by weight.
Litharge,	. . .	150 " "
Slag,	. . .	20 " "

The products of different mines will necessarily vary, and the proportions to give a clean, well-oxidized slag can easily be determined by experiment on a small scale with assay crucible tests. Mr. Tavener, however, mentions that considerably less assay, or other slag, will effect a good fusion in a reverberating furnace than in a crucible, and he states that it would be safe to use 30 per cent. less slag when smelting in the reverberatory than was found necessary in the crucible trial.

The quantity of litharge should be so proportioned that the lead bullion should not carry more than 8 per cent. of gold, or 10 per cent. at the maximum. It was found preferable to make a larger quantity of lead than to have it too rich in gold contents.

No reducer is used with the fine zinc, which is relied on to reduce sufficient lead, leaving an excess of litharge in the slag to ensure success. With the gold slimes charge, 1 per cent. of sawdust is added on the weight of litharge, but if a larger proportion of litharge has been used, then from $1\frac{1}{2}$ to 2 per cent. of sawdust is necessary.

The furnace work is described as follows:—

"When the entire charge of zinc and slime is in the furnace, it is banked up from the sides to the centre so as to avoid the possibility of particles remaining on the sides above the slag level as the charge reduces and settles down. A covering of litharge is spread over the surface, and on this again a light covering of easily fusible slag is spread. The furnace is charged the day previous to smelting, and one of the night-shift men lights a slow fire about 3 a.m., which serves to dry the charge. At 5 a.m. the damper is opened and the fire urged, and in half an hour the furnace is at a smelting heat. By 9 or 10 a.m. the charge is reduced, then sweepings from cyanide works, smelting room, or

any slag requiring re-smelting, is added and is quickly absorbed in the molten bath. When all this has been fed in and melted, and the slag become fluid, it is well stirred with a rabble, and sawdust is thrown in to reduce the excess of litharge in slag. This operation is repeated until the slag, which remains on the rabble when withdrawn from the furnace, is judged by its appearance to be clean. The slag is now run off into pots through the slag-door, the level of which is 4 in. above the centre of the lead bath—a bath of 12,000 oz. of lead bullion almost occupies this space.

“Before filling the furnace, the slag-door is built up about 12 in. by placing flat cast-iron plates, $\frac{1}{2}$ in. thick, bedded in fire-clay, one on top of another, and in front of these plates a bank of fire-clay is also made. In order to run the slag off, all that is necessary is to break away this bank, plate by plate, and so allow the slag to flow over into the pot. When the pot is full it is wheeled away and another is put in its place. The filled pot is run outside, and after standing a minute or two, is tapped, and the molten slag allowed to run out on the ground to cool; that which remains on the sides and the bottom of pot is brought back for further use. When no more slag will flow from the furnaces owing to the bath being down to the level of the slag-door, it is waved off by rabbling. At first sight it would appear difficult to draw this remaining slag off without dragging out some lead, but a very little practice enables it to be done so closely that there is little but a thin skimming of slag remaining. In the event of a little lead being pulled out into the pot, it is recovered from slag pots. It is for this reason that the pots are tapped about 2 in. from the bottom. By opening the fire door this last skim on the lead bath quickly thickens. A shovelful of lime is thrown in to assist. This skim is easily pulled off, and of course is held over until next smelt. By this means a clean surface of lead is exposed, and any zinc present would be quickly got rid of, for at this stage the lead is at a bright red heat, and the free access of air due to the open fire door quickly oxidizes it. So far, lead recovered by this method has always been clean and soft, a proof that no zinc could be present, since one per cent. of zinc gives lead a distinct silvery colour, and makes it so hard that it cannot be rolled. The lead bullion is tapped by driving a $\frac{1}{2}$ in. steel bar, tapered to a point, into the tap-hole, which is closed with a fire-clay plug. The lead is run into an iron trough, which conveys it to the moulds placed together on the floor.”

Before tapping the furnace the lead-bath is well stirred, and a sample is taken out with a ladle and granulated.

The cupelling or refining of the lead bullion is next described, with particular details and some useful hints on the making of the

bone-ash test and regulation of blast. The process differs only in minor details from the usual operation of lead-refining, and need not be specialized in this work.

The costs at the Bonanza, Limited, for stores for cupelling and smelting for four months (June–Sept. 1902) were as follows :—

Coal, ...	£82 13 1			
Coke, ...	9 8 6			
Fire-clay, ...	3 0 0			
Fire-bricks and slabs, ...	17 15 6			
Paper bags, ...	1 17 6			
Lead foil, ...	3 0 8			
Bar iron, ...	0 3 2			
Crucibles and liners, ...	1 15 0			
Caustic potash, ...	0 12 0			
Bone-ash, ...	10 0 0			
Sundries, ...	1 5 2			
				Cost per fine oz. recovered.
	£131 10 7	=	2·463d.	
Lead loss estimated at 12% on 15,269 lbs. = 1832 lbs., say 1 ton at £20, ...		20 0 0	=	373d.
		£151 10 7	=	<u>2·837d.</u>

Gold output in period, 12,810 oz., fine.

It is claimed that lead-smelting recovers a larger amount of gold than the sulphuric acid method, and this claim seems to be justified by several large experiments on equal weights of identical gold slimes by both processes. In six trials the lead process gave 10·5 per cent. higher recovery than the acid treatment. These results are probably much higher than could be obtained in continuous working; but it is evident that if only one or two per cent. more gold can be recovered, the lead-process has fully established the claims of Mr. Taverer.

In the discussion which followed the reading of Mr. Taverer's paper, Mr. D. J. Williams suggested that the probable loss of gold by volatilization in the cupellation, caused by the presence of such a volatile metal as zinc, might be avoided by first dissolving the zinc-gold slimes in sulphuric acid, washing, drying, and then subjecting to lead-smelting. Mr. Taverer agreed with this proposed modification, and stated that he would rather receive at the furnace acid-treated precipitate, which would render the work easier and less troublesome. He further stated that with zinc present, a

certain amount of experience and skill was required, while with acid-treated material little or no experience was needed.*

That lead-smelting recovers a higher percentage of gold than the acid-treatment seems to be established on pretty conclusive evidence, but it is not quite so easy to point to the source of gold thus recovered; that is, to show where the loss takes place in acid-treatment.

Zinc Dust Precipitation.—The method of application is to agitate a certain quantity of the dust with the gold-containing cyanide solution in vats, allow the precipitate to settle, and decant the clear solution.

During the past year or two this method of precipitation has been adopted at a number of American mills, among which may be mentioned the Homestake Mining Company, South Dakota; the Montana Mining Company, Montana; the De La Mar Mining Company, of Nevada; and the Consolidated Mercur Company, of Utah.

At the Homestake Company's cyanide works "Precipitation is carried on by means of zinc dust in five tanks 25 ft. in diameter and 20 ft. deep, built of California red-wood. The bottoms are inclined to one side, where a small sump is provided in order to drain the tanks perfectly. No gold storage solution tanks are used, the solutions draining from the leaching vats directly into the precipitating vats. The precipitation vats are pumped out by duplex Prescott pumps, the solution going to filter presses made in the Homestake shops. The pulp treated by the mill is of a highly siliceous nature, often however carrying some hornblende, the sulphides being mainly pyrite and pyrrhotite. The extraction made is in the neighbourhood of 90 per cent., the cost being between 45 and 50 cents per ton of material treated, probably nearer the latter figure."†

In America the price of zinc dust varies from 25s. to 29s. per cwt., and the amount of dust used for precipitation varies from 6 to 9 oz. per ton of solution.

Charcoal Precipitation.—At many cyanide plants in Victoria charcoal is being used to precipitate the gold from cyanide solutions. The solution is passed through a series of barrels packed with finely broken charcoal, on which the gold is deposited. The charcoal is afterwards burnt to an ash, and the ash fluxed. The process is too slow and cumbersome to recommend itself for use in large plants where hundreds of tons of solution have to be handled in the twenty-four hours.

* *Jour. Chem. and Met. Soc. S.A.*, Jan. 1903.

† C. H. Fulton, *The Engineering and Mining Journal*, June 4, 1902.

CHAPTER XI.

THE APPLICATION OF THE PROCESS IN DIFFERENT COUNTRIES.

SOUTH AFRICA.

At the Witwatersrand Goldfields the cyanide process has been conducted on a more extensive scale than elsewhere. The ore there is principally a pyritic, silicious quartz-conglomerate, consisting of rounded or sub-angular pebbles of bluish-grey quartz embedded in a quartzose matrix. The pyrites occur in varying proportions in different mines, but the average is probably not less than 2 per cent. The gold does not exist in the quartz pebbles, but occurs disseminated throughout the matrix or in the iron pyrites.

This ore is locally known as "banket," or almond rock. It is comparatively hard and somewhat splintery, and often contains a small proportion of corundum and clay, which renders it tough and hard, and forms slimy products during the crushing.

Throughout these goldfields the universal practice at present is wet-crushing with Californian stamps, copper-plate amalgamation, concentration or classification of tailings, cyanide treatment of tailings, and chlorination or cyanide treatment of concentrates.

At most of the batteries a 30-mesh screen is used, but in a few cases a finer or coarser mesh is adopted.

The main features of the cyanide treatment at the Witwatersrand Goldfields are much the same as those practised in America and Australia. The general working details are given below in tabulated form. They are the same at all the cyanide plants, with minor differences according to the individual fancy of the chemist or metallurgist in charge of the operations, and the requirements of the ore or tailings.

	Hours.
Filling vats, ...	12
Preliminary alkali or water wash, if necessary, .	12
Strong solution, 0·2% to 0·4% solution, $\frac{1}{4}$ to $\frac{1}{3}$ of ore—	
In contact with tailings, ...	12
Percolating, ...	12
Dry or air-leaching, ...	4
Weak solution, 0·15% to 0·2% solution, about $\frac{1}{4}$ of ore, ...	12
Dry or air-leaching, ...	4
Two weak cyanide washes, 0·05% to 0·1% washes, each about $\frac{1}{8}$ of ore, ...	12
Two clean water washes, each about $\frac{1}{8}$ of ore, ...	12
Discharging vat, ...	8
Total, ...	100

The total quantity of solution used, including the water washes, is about equal to the weight of the ore. The quantity of strong solution used varies according as a preliminary washing with a dilute solution has been employed or not. In the former case it is about 25 per cent. of the weight of the ore, and in the latter case about 40 per cent.

The percolation vats are charged with tailings to within a few inches of the top, and their surface is levelled. The strong cyanide solution is then allowed to penetrate the tailings until they are covered. The contents of the vat settle some inches, the amount of shrinkage depending on the depth of the vat and the percentage of moisture in the tailings.

The value of the tailings varies from 12s. to 20s. per ton, and the actual extraction amounts to 70 or 75 per cent., at a cost varying from 4s. to 10s. per ton, according to the size of the plant.

The Jumpers Deep cyanide plant and slime works are among the newest on the Rand, and represent the most advanced and up-to-date practice. The cyanide works of the Waihi Company at their new mill at Waikino are modelled on the latest South African practice and experience; and as they are fully described in the next pages, it will be unnecessary to give further details of South African works and practice, all of which are lucidly described by Mr. John Yates in his recent work on metallurgical engineering on the Rand.*

* John Yates, "Present Day Metallurgical Engineering on the Rand," 1898, *The Mining Journal*, London.

New Kleinfontein Mine.—In a paper read before the Institution of Mining and Metallurgy, London,* Mr. F. Cardwell Pengilly gives some interesting details of the successful treatment of tailings by the "direct filling process." The plant, he says, consists of 2 slime separators, 19 treatment vats, 4 extraction boxes, and 9 solution tanks. The slime separators are of the ordinary spitzkasten or pointed box style, 6 ft. square at the top and 6 ft. deep. Of the 19 treatment vats, 13 are of 200 tons capacity, and 6 of 130 tons capacity each. Around the top of each vat is a launder, which carries away the overflow into the slimes race. The 9 solution tanks have a total capacity of 636 tons of solution, and each is connected with the three centrifugal pumps worked from a motor that pumps the weak, medium, or strong solutions on to the treatment vats. The pulp or tailings, after leaving the battery plates, is lifted by means of a tailings wheel into a launder, by which it is conveyed to the slime separators. The slimes from the first separator pass into the second, whereby a quantity of fine sand is saved that otherwise would flow away. About 15 per cent. of slimes are eliminated in the separators, and the remaining pulp is run through launders into the treatment vats, which are previously filled with water. In the course of filling the vat, another 10 per cent. of slimes overflows into the slimes race, so that the resulting tailings to be treated in the vat contain but a small proportion of slimes.

The vat is filled with tailings (sand) to within 1 foot of the top, when the stream is diverted into another vat.

Method of Treatment.—Each vat, after being drained of water, is treated with an alkaline solution to neutralize the free acid in the tailings formed by the decomposition of pyritic ores. As soon as the solution draining away is slightly alkaline, the treatment by cyanide solution is commenced. Various strengths of solution are pumped into the vat, each vat receiving during its course of treatment from 200 to 275 tons of cyanide solution. The length of treatment is six days, and the amount of cyanide used is 6 lbs. per ton of tailings.

The following table shows the exact method of procedure adopted in the works in the treatment of a vat of tailings:—

* *Trans. Inst. Min. and Met. London*, vol. vi. p. 113, 1898.

VAT I. CHARGE I. CAPACITY 207 TONS.

Date filling.	Time.	Test.	Tons.	KCy Solution.
October.				
Wed. 6th	1 p.m.	...	Filling.	
"	2 a.m.	...	Filled leaching dry.	
"	4 "	Acid	Alkaline wash, 20 lb.; NaHO used, 20 lb.; $\frac{1}{2}$ bag lime.	Tons.
"	9 "	Alkaline	Preliminary solution, 0'16; KCy, 20 tons.	20
"	2 p.m.	...	0'01 changed to weak box.	
"	2 "	...	1st strong, 30 tons 0'3 KCy; 3 p.m., 20 tons; 7 p.m., 6 tons.	56
"	9 "	0'09	Standing under solution for 3 hours.	
"	12 p.m.	...	Started leaching until 12 noon.	
Thurs. 7th	1 a.m.	0'1	Changed to medium box, 3 a.m., 1'3; 6 a.m., 1'6; 9 a.m., 0'18.	
"	12 noon	0'2	Changed to strong precipitating box.	
"	12 "	...	2' strong, 30 tons; 0'3 KCy, 3 p.m., 10 tons; 6 p.m., 15 tons; 8 p.m., 1 ton.	56
"	8 p.m.	0'28	Standing under solution for 3 hours. Started leaching until 8 a.m.	
Frid. 8th	8 a.m.	0'26	Medium, 20 tons; 12 noon, 5 tons; 4 p.m., 8 tons; 8 p.m., 8 tons.	36
"	9 p.m.	0'2	Changed to medium; 12 p.m., 8 tons; 5 a.m., 10 tons; 12 noon, 7 tons.	25
Sat. 9th	3 "	1'8	4 p.m., 10 tons; 8 p.m., 7 tons; 4 a.m., 12 tons; 8 a.m., 5 tons.	34
Sun. 10th	6 "	1'4	Weak, 5 tons; 12 p.m., 5 tons; 6 a.m., 5 tons; 12 noon, 5 tons.	20
Mon. 11th	6 a.m.	0'1	Changed over to weak.	
"	6 p.m.	0'9	5 tons; 12 p.m., 5 tons; 7 p.m., 5 tons; 12 p.m., 4 tons.	19
Tues. 12th	6 a.m.	...	Leaching dry.	
Wed. 13th	6 "	...	Discharging.	
				266

Amount of KCy used, 5'9 lbs. per ton.

Precipitation of Gold in the Extractor Boxes.—The precipitation of gold from the solution coming from the treatment vats is effected, he says, by the means of zinc shavings. A certain portion of the ore coming from the mine is of a rebellious nature, containing sufficient foreign metals to retard, if not prevent, the precipitation of gold by zinc. To counteract the effects of these foreign metals, it has been proved efficient in practice on these fields to set up a lead couple, and at these works this is effected by dipping the zinc shavings in a weak solution of acetate of lead, preparatory to being placed in the extractor boxes.

In all cases it is found that zinc containing from 1 to 2 per cent. of lead gives the most satisfactory precipitation. By the use of acetate of lead a perfect precipitation is brought about; only traces of gold remaining in the solution after passing through the extractor boxes.

The actual working costs amount to 1s. 11d. per ton, including general charges and maintenance.

It has been shown in America and Western Australia that with certain classes of ore and certain local conditions, it may be more advantageous to dry-crush than wet-crush preparatory to treatment. In South Africa wet-crushing is universal. Dry-crushing in that country has never been viewed with much favour, nevertheless the experiments made by Mr. Franklin White at the Luipaards Wei Estate mine are interesting and instructive.*

The tanks used were made of steel, 25 ft. in diameter and 8 ft. deep, with the usual filter bottoms and bottom discharge doors. Three tanks were placed close to the mill and mine in a row some distance away, and a little below. The ore was treated a few days in the upper tanks and then transferred to the lower, a double treatment being thus obtained. Mr. White does not consider double treatment to be of advantage in dealing with dry-crushed material, as the ore is properly mixed in the first instance, and there is abundance of air entangled in the dry sand. He is rather of the opinion that when the damp ore is transferred to the second row of tanks, there is a tendency to pack closer and to retard filtration.

Lime was added to the ore at the rock-breaker floor, in proportions varying from 1 lb. to 2½ lbs. per ton. By this means it was thoroughly mixed in the different machines, and any lumps were broken up.

The actual cyanide treatment differed a little from the ordinary procedure, and followed lines first suggested to the author by Mr. W. R. Feldtmann.

* *Trans. Inst. Min. and Met. London*, vol. vii. p. 124, 1899.

As soon as the tank in the first row was filled, a strong solution, 0.25 per cent. KCy, followed by two others, the last being 0.15 per cent., was pumped on to it. The time given to this treatment would be sixty-six to seventy hours. The solutions were not allowed to stand, but were drained off when the tank was once filled. The object of this was to allow fresh air to obtain access to the mass of damp sand. Each solution would be about 27 tons to the tank of ore (165 to 170 tons).

The ore was then transferred to the second row of tanks, having lost about 67 per cent. of its original assay value in this short time. It is probable that the abundance of air entangled in the dry sand, as compared with what would be held in a tank of sand settled with water, materially assisted the solution of the gold. Also the finer grains of the free gold would be quickly taken up.

A solution of 20 to 25 tons, not exceeding 0.20 per cent. KCy, was then pumped on to the transferred sands and drained off, the mass being allowed to remain damp for about ninety-six hours, when weaker solutions (0.15 per cent. to 0.10 per cent.) were used in continuous washes, making up a total of 75 tons per tank (second treatment). A water wash of 20 or 30 tons completed this part of the process, which would last some 275 hours.

A careful series of moisture tests and measurements of solution sumps during the treatment of four tanks (680 tons) showed that the total loss of liquid in the treatment was 96 tons, or 24 tons per tank. The moisture in the discharged residues averaged 12.3 per cent., or, say, 20 tons per tank; the remaining 4 tons would be represented by evaporation from surfaces of sumps and tanks and by leakages. As the ore contained about 3 tons of water per tank, in the form of moisture when delivered from the mill storage bin, the actual consumption of fresh liquid equals 21 tons per tank, or $\frac{1}{8}$ ton (25 gallons) per ton of ore.

The solutions running from the first row of tanks carried from 13 dwt. to 32 dwt. per ton; those from the second row 3 dwt. to 4 dwt.; and the final wash 0.8 dwt. to $2\frac{1}{2}$ dwt.

Summary of Cyanide Costs.

TRIAL CRUSHING (COARSE).		s.	d.
Tanks and extractor house work,	per ton	2	3.30
Clean up and smelting, etc.,	"	0	5.15
Discharging residues,	"	0	8.36
Sundries,	"	0	1.05
Total		3	5.86
Actual extraction 68.91 per cwt.			

NEW ZEALAND.*

The principal gold-bearing formation of the Hauraki Goldfields is of volcanic origin, consisting of a great accumulation of andesitic lavas, tuffs, breccias, and agglomerates of lower tertiary age. These rocks everywhere bear evidence of having been subjected to the prolonged leaching action of thermal waters, doubtless accompanied by steam and acid vapours. They are found in all stages of decomposition or alteration; and in many mines the hard blue andesite can be seen to pass by a series of almost insensible gradations into a soft, or fairly hard, greyish-yellow or blue altered rock, to which the distinctive name propylite has been applied.

It is in this altered andesite that the veins yielding payable ore occur. The veins vary from a few inches to 40 ft. in width, but in linear extension they can seldom be traced for any considerable distance. The ore values are also irregular, and in no case, as yet, is the same vein or lode worked with payable results in two adjoining mines.

Cyaniding Ores.—In the southern portion of the Hauraki peninsula, the pay-ores consist of whitish-grey chalcedonic or cryptocrystalline quartz, often possessing a wavy, banded structure of alternating layers of grey and blue flinty quartz. They are comparatively free from base sulphides.

The gold is about '645 fine, and usually associated with silver sub-sulphide (Ag_2S) in varying proportions. It is generally extremely finely divided, being seldom visible to the eye, and in the great bulk of the Waihi ore it is impossible to raise even a colour by panning.

Prior to the introduction of the cyanide process, these ores were treated by dry-crushing and hot pan-amalgamation with chemicals, by which a recovery of 65 per cent. was effected.

When cyanide treatment was adopted, dry-crushing was naturally continued at the different mills, the dry pulverized material being charged into shallow vats and treated directly by cyanide. From 65 per cent. by pan-amalgamation, the recovery rose at a bound to 85 per cent., and in some cases to 90 per cent., and the results were considered so satisfactory that no further improvement was considered possible.

In a few years, however, it became apparent that dry-crushing possessed many disadvantages compared with wet-crushing, the principal being the cost of drying the ore, the low duty of the

* Excerpt from paper by author read at California Meeting, September 1899, American Institute of Mining Engineers.

stamps, and the large number of vats required for leaching. In 1897 mine owners began to turn their attention to wet-crushing, and one by one, since the beginning of 1898, the different mills have been adopting wet-crushing, until at the present time dry-crushing is the exception and not the rule, as it was two years ago.

Wet-crushing Practice.—A. For ores containing a large proportion of free, easily amalgamable gold, with a small proportion of fine or "float gold" and silver sulphide, the mill practice is:—

- (a.) Crushing with water.
- (b.) Plate amalgamation.
- (c.) Spitzlute separation of sands and slimes.
- (d.) Cyanide treatment of sands and slimes by ordinary percolation.

A typical example of an ore of this class is that of the Kauri Gold Estates at Opitonui, where a new 40-stamp mill has just started. The sands and heavy slimes are subjected to the "double" cyanide treatment, but it is doubtful if the additional saving will pay for the extra labour involved. So far no provision has been made for the treatment of the fine slimes. If they are worth it, they will probably be treated by agitation and decanting.

B. For a clean ore, almost identical with that described above, but containing a small proportion of free amalgamable gold, and a large proportion of fine cyanide gold, with little or no slimes, the method of treatment at the Crown Mines is:—

- (a.) Crushing with cyanide solution.
- (b.) Direct cyanide treatment of mixed sands and slimes by percolation.
- (c.) Plate amalgamation of free gold.

With an ore so exceptionally free from slimes, it seems that the order of treatment could be reversed with advantage, both as regards stamp duty and efficiency of amalgamation on the plates.

The Crown Mines Company was the first to adopt wet-crushing for these gold and silver-bearing chalcedonic ores, which occurred in 1897, and much credit is due to Mr. F. R. W. Daw, the superintendent, for the successful inauguration of the method.

The ore is hard and splintery, clear and pure from all impurities, and, unlike most of the ores from the neighbouring mines, contains little or no silver except what is alloyed with the gold. It is crushed in the Company's 60-stamp mill with cyanide solution in the mortars, about $2\frac{1}{2}$ tons of solution being used to 1 ton of ore. A 25-mesh screen is used, and the duty per stamp is about 2 tons per day. The slimes formed in crushing are said to amount to less than 5 per cent.

The monthly output is about 2750 tons. For the month of March 2916 tons were crushed, yielding bullion valued at £5797. The bullion was worth a fraction under £2 per ounce.

The cyanide plant consists of twenty-eight leaching vats, each $22\frac{1}{2}$ ft. in diameter and 4 ft. deep. Much reticence is maintained as to the exact treatment, but the main features are understood to be as follows:—

The whole of the pulp from the 60-stamps is conducted by a launder to one vat, and allowed to discharge into the centre until the vat is about half full. The pulp is then diverted to another vat, which is allowed to fill in the same manner. The mixed sands and slimes in the first vat are allowed to settle for an hour or two, after which the fairly clean top solution is syphoned off into a collecting tank, whence it is pumped up to two elevated tanks, from which the solution for the stamps is supplied. The pulp is again diverted into the first vat until the charge is complete. After settlement, the top clear solution is again drawn off. In this way three or four vats may be in course of filling at the same time.

The settlement of the slimes is effected without the aid of lime, by allowing the solution to percolate from the bottom of the vat during the periods of filling and settlement. The downward tendency of the currents promoted by the draining from below is said to cause the settlement of the finest matter within a reasonable time. This is a point that should be noted by cyaniders troubled with slimy products.

The mixed sands and slimes are treated by percolation in the ordinary way. The depth of each charge is about 30 inches, and the weight 40 tons. The strong cyanide solution is allowed to percolate from thirty to forty hours, while the weak cyanide and water washes are drawn off by the aid of an air-pump.

The syphon used in the sand vats consists of a length of $2\frac{1}{2}$ in. rubber hose, to one end of which is attached a short length of wooden batten to keep it on the surface of the solution. The other end is fixed, inside the vat, to a short iron pipe passing through the side about 18 inches above the filter-cloth.

In the extractor-room there are five precipitation boxes of the ordinary pattern, divided into compartments by baffle boards; and four zinc towers, consisting of wooden boxes about 6 ft. high and 30 in. square, set on end and connected in a series like charcoal towers. The solutions flow upwards through the zinc turnings, the overflow being conducted in a pipe to the bottom of the next tower, and so on to the last.

The sands are sluiced out of the vats over a wide expanse of

amalgamated copper-plates, which catch a certain proportion of the free gold.

The actual recovery from all sources is said to vary from 84 to 87 per cent., but the costs are not obtainable.

C. For ores containing some easily amalgamable gold, and fine gold associated with pyrites and silver sulphides, the treatment used is :—

- (a.) Crushing with water.
- (b.) Plate amalgamation.
- (c.) Spitzlute separation of fine slimes, if necessary.
- (d.) Vanner concentration of sulphurets.
- (e.) Cyanide treatment of sands by percolation.
- (f.) Cyanide treatment of slimes by agitation and decanting.
- (g.) Cyanide treatment of concentrates by agitation.

The practice at the Woodstock mill is a typical example of this treatment.

The ore is a chalcedonic and finely crystalline quartz, containing a small proportion of clayey matter and a little pyrites. It is stained a greyish and blackish-brown colour through the presence of iron and manganese oxides.

At the Company's 40-stamp mill the monthly output is about 1100 tons, the stamp duty being slightly under 2 tons of 2240 lbs. per day. For the June month, 1901, 1000 tons of ore were crushed for a return of £1362, which is equal to a value of £1, 7s. 6·68d. per ton. The value of the bullion varies from 8s. to 12s. per ounce, being principally composed of silver.

The ore is crushed with water and passed over amalgamated copper-plates, from the end of which the pulp is raised by a wheel elevator to a spitzlute. The slimes from the spitzlute are conducted to a slime tank, while the sands carrying some heavy slimes are passed over vanners, which collect about 1 per cent. of rich concentrates.

The vanner tailings, composed principally of sands and heavy slimes, are led to the leaching vats, which are provided with automatic distributors. The construction of the distributors is of the simplest character, being similar to those formerly used at Waihi. They consist of a central wooden box, pivoted on a wooden pillar fixed in the centre of the vat, and from which extend seven narrow wooden launders or arms of light make and different lengths, so as to effect an even distribution of the pulp. At the end of each arm there is fixed a piece of sheet zinc to divert the stream to one side.

The whole of the pulp from the vanner ends is collected in one stream and diverted into one vat at a time until the charge is

filled. During the filling the overflow carries the lighter slimes into the slime vats.

The sands and heavy slimes are treated with cyanide by ordinary percolation.

The slimes from the spitzlute, and those from the sand vats, are agitated with cyanide in vats provided with slowly revolving arms. When the gold is dissolved lime is added, and the slimes are allowed to settle, after which the clear solution is decanted off. The slimes are washed by agitating with successive portions of water and decanting.

The concentrates have a value of £30 to £40 per ton, a large proportion of the value being in silver sulphide. They are treated by agitation with a 4 per cent. solution of cyanide for thirty-six hours. Two pounds of lime are added to every ton of concentrates. The charge weighs about $1\frac{1}{2}$ tons. Mr. F. Rich, the superintendent, who adopted the present treatment, informed the author that the recovery varied from 90 to 94 per cent., at a cost of 18s. per ton for labour and material.

The recovery by cyanide from all sources per ton of ore milled is said to vary from 82 to 86 per cent., at a cost of 4s. 9d.

D. For very slimy ores containing very little easily amalgamable gold, and a large proportion of extremely fine gold besides the usual silver sulphides, the treatment is as follows :—

- (a.) Crushing with cyanide solution.
- (b.) Spitzlute separation of sands and slimes.
- (c.) Treatment of sands by percolation.
- (d.) Treatment of slimes by agitation and decanting.

This method of treatment is subject to various modifications as regards mechanical appliances and methods of application, but the general principles are everywhere the same.

The procedure at the Waitekauri 40-stamp mill is as follows :—

The ore, which contains a good deal of oxidized products, is crushed with cyanide solution in the mortars. From the screens the pulp is conducted direct to the sand vats, into which it is distributed by means of revolving wooden box launders actuated from a secondary shaft. The slimes, of which there are about 33 per cent., are allowed to drain into a collecting vat, flowing through a pipe fixed in the inside of the vat. This pipe has a movable joint, and is raised by a screw as the pulp accumulates in the vat.

The collecting vat is provided with revolving arms which keep the fine slimy pulp from settling. From this vat the slimes are pumped into the slime leaching vats, which are provided with a double set of slowly revolving arms, the lower ones having rakes

on them and the upper ones loose pieces of sacking which drag through the pulp. In these vats the slimes are treated by agitation and decanting, lime being added with each wash to facilitate settlement.

The sands are treated by ordinary percolation with first a 0.5 per cent. solution of cyanide, and then the weak and water washes.

There are 12 sand vats, 14 slime vats, and 2 slime collecting vats each 22.5 ft. in diameter and 4 ft. deep. The monthly output of the 40-stamps is about 2200 tons of 2240 lbs. The June monthly output was larger than usual, being 2543 tons, yielding 7220 ozs. of bullion valued at £6773, 0s. 6d., equal to a value of 18s. 9d. per ounce. The actual recovery is said to be 90 per cent., at a cost of 5s. 6d. per ton.

The exceptionally large proportion of slimes in this ore rendered the adoption of wet-crushing a knotty and difficult problem. The increased output, higher extraction, and lower costs are proofs enough of the success of the change from dry-crushing which was effected under the supervision of Mr. G. Davey, the superintendent, without hitch, or decrease in the monthly output, a matter of no little moment in these days of heavily capitalized public companies.

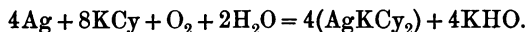
Concluding Remarks.—Among the points most likely to attract the notice of cyaniders are the low stamp duty and the heavy consumption of cyanide.

The low stamp duty of the New Zealand mills has often been a subject of discussion, but no satisfactory explanation has yet been advanced. In the opinion of the author, it is due to the circumstance that the mills having been designed and erected in the first place for dry-crushing, the mortars are too narrow and restricted to give good results by wet-crushing, and until these are replaced by new mortars, specially designed for wet-crushing, it seems hopeless to look for better results. The advantages of a stamp duty of 4 or 5 tons per day instead of 2 tons are too obvious to require enumeration.

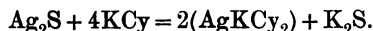
With reference to the large consumption of cyanide, it is well known that silver in all its forms requires a stronger solution to effect its dissolution than gold; and in the Hauraki Goldfields the large consumption of cyanide is due to the presence of silver sulphide (principally Ag_2S), which is generally worth saving, and to the circumstance that the free gold is alloyed with about one-third its weight of silver.

According to Elsner's equation for the dissolution of gold by potassium cyanide, 4 lbs. of cyanide should dissolve 100 ozs. of gold, but in practice it is found that it takes nearly forty times

that quantity. To dissolve 100 ozs. of silver would require 7·5 lbs. of cyanide, according to the equation :—



For the dissolution of 100 ozs. of silver existing as the sub-sulphide (Ag_2S), 7·01 lbs. of cyanide would be required by the following equation :—



The potassium sulphide resulting from the dissolution of silver sulphide also tends to cause a loss of cyanide. It has been shown by Crosse and others that a trace of alkaline sulphide in cyanide solutions does not act injuriously, but the large quantity of K_2S liberated in the treatment of the silver-bearing ores of the Hauraki Goldfields must cause the precipitation of a portion of the dissolved gold in the vats. Much of this precipitated gold is doubtless redissolved by the excess of free cyanide present in the solutions, but it always requires this excess to obtain adequate extractions, thus necessitating the use of comparatively strong solutions.

One of the most perplexing features connected with the treatment of these ores is the constantly varying proportion of silver, which necessitates the use of solutions of varying strength to obtain adequate extractions, thus adding another source of anxiety to the many worries which the use of cyanide entails on even the successful cyanider.

At many of the Hauraki mines, cyanide treatment was adopted by the owners on the author's recommendation, but only after he had made a careful investigation of the constituents of the ore, and repeated trials on a working scale, at the Government Metallurgical Works at the Thames. In other cases, the necessary experimental trials were made by the author's assistants at the mine. In no case was the process adopted until success had been assured, a precaution which doubtless has been a potent factor in promoting the popularity of cyanide treatment in this country.

The ores of Te Aroha and Monowai are generally very refractory, containing free milling gold, mostly very fine, associated with sulphides of silver, iron, copper, lead, zinc, and often mercury. Many attempts have been made to treat them by cyanide, but without success, and, so far as our present knowledge goes, it is doubtful if they can ever be treated successfully in the raw state by that process.

For the treatment of cupriferous ores and concentrates from the Jubilee, Sylvia, and Monowai mines, which could not be treated successfully by ordinary cyaniding, the author obtained good results by first subjecting the ore to a chloridizing roast, and

then leaching out the copper chlorides with water. After an alkaline and water wash, the gold and silver contents were extracted by cyanide by ordinary percolation. During the roasting the silver sulphides present were chloridized, the chloride being easily dissolved by cyanide.

From a large parcel of Monowai ore, 92 per cent. of the gold and 85 per cent. of the silver were extracted, the composition of the ore being (F. B. Allen, M.A., B.Sc.) :—

Insoluble gangue, ...	90.15
Copper pyrites, ...	3.78
Iron pyrites, ...	4.40
Galena, ...	0.25
Zinc-blende, ...	0.26
Alumina, ...	0.13
Water and loss, ...	1.03
	100.00

The bullion contents of this ore were :—Gold, 1 oz. 5 dwt.; silver, 14 oz., per ton.

The Moanataiari Co.'s Works, situated at Thames, are of recent construction, and in point of completeness and modernness of equipment, among the finest in New Zealand. The plant consists of a 60-head battery, two rock-breakers, grizzlies, automatic feeders, 24 vanners, 21 Berdan pans, a complete cyanide plant for treating vanner concentrates, 9 Cornish buddles for concentrating the tailings from the vanners, and all up-to-date appliances for assaying and retorting. The cyanide works consist of three steel vats, 20 ft. in diameter and 7 ft. deep, each provided with two bottom-discharge doors, with a capacity each of 200 tons of concentrates; two zinc extractors, each 15 ft. 6 in. in length; and three concrete sumps, 50 × 11 × 6 ft. over all. Wet-crushing, concentration, and the cyanidation of the concentrates are the interesting features of the practice at these works, which were designed by the author and erected under his supervision. The cyaniding of concentrates is comparatively new in New Zealand.

Talisman Mine.—In the Ohinemuri Goldfields, the practice of dry-crushing and direct cyanide treatment has been superseded by wet-crushing, concentration, and cyanide treatment of sands, slimes, and in some cases the concentrates. At the Talisman mill, the ore was conveyed from the mine by an aerial tramway being dumped on to a grizzly, which passed the coarse stuff to a Blake Marsden crusher, whence the ore passed to a revolving

drying furnace. The stamps weighed 1000 lbs., and the mortars were provided with back and front discharge.

The pulverized ore was elevated to an ore-bin, from which it was carried to the cyanide vats by a long line of revolving screw-conveyors. The mechanical drier was a very efficient machine, drying about 12 tons of ore for every ton of firewood consumed.

This company's dry-crushing mill has been dismantled, to give place to a new 50-stamp wet-crushing plant. Wire plate-amalgamation, vanner concentration, separation of slimes, cyaniding of sands and slimes, the latter by agitation and decantation, has been introduced.

Waihi Mine.—The ore in the upper levels of the celebrated Martha lode at the Waihi mine is typical of most of the ores in this district. It consists principally of hard, splintery, whitish-grey chalcedonic and crypto-crystalline quartz, often possessing a banded and wavy structure. It is perfectly free from all base metallic sulphides, and the amount of iron oxides present is so small that when roasted and pulverized the colour of the dust is pinkish-grey.

The value varies from £4 to £5 per ton, the precious metals existing in the proportion of about 3 oz. silver to 1 oz. gold. The free gold is alloyed with about 35 per cent. of silver, being valued at about 53s. per oz. The greater proportion of the silver exists in the form of the bluish-grey sub-sulphide known as argentite. In the surface levels, thin leafy plates of gold were not infrequently seen adhering to the surface of large cuboidal masses of quartz, but in the lower levels a colour is rarely seen, the gold existing in an extremely fine state of subdivision. Such an ore is theoretically perfect for cyanide treatment, and actual experience has proved it to be so. By the old stamp battery, and copper-plate amalgamation, the recovery amounted to only some 4 dwts. per ton, equal to about 15 per cent. of the value. By dry-crushing and pan-amalgamation the extraction was raised to 60 per cent., but at present the actual extraction by the cyanide process amounts to over 90 per cent. of the assay value.

The following particulars of the dry method of treatment formerly in use at the Waihi mills were supplied by Mr. H. P. Barry, the general manager, for the annual report of the New Zealand Mines Department for 1894:—

Drying the Ore.—The ore is trucked to the drying kilns which consist of open circular holes excavated in the solid rock, their dimensions being 37 ft. in depth and 20 ft. in diameter at the top, and tapering somewhat at the bottom. Each kiln is capable of holding 100 tons of ore at a charge. The lower part is lined with bricks, and finished off with a brick arch, having a door

and an iron chute for discharging the dried ore into trucks, which have access to the kiln by means of a tunnel cut in the rock.

Charging the Kilns.—The kilns are charged with alternate layers of wood and ore, the layers of wood being about 5 ft. apart. When the kiln is fully charged, the wood is lighted, and, after it is all burned up, about half the charge, that is about 50 tons, is withdrawn and another 50 tons of raw ore, together with the necessary wood, are placed on the top. After this about 50 tons of ore are withdrawn every third day, while a similar quantity of raw ore and wood is added.

The method of drying the ore is very expensive, as one ton of wood will only dry about three tons of ore. The cost of firewood at the Waihi big mill is 2s. per ton of ore dried, and the total cost of drying, including labour, is 2s. 6d. per ton.

Crushing and Pulverizing.—From the kilns, the dried ore is trucked to the rock-breakers, whence it passes by gravitation to the self ore-feeders. The pulverizing machinery consists of a 90-stamp battery and an Otis ball-mill, having a capacity of about 10 stamps. The ore is passed through a 40-mesh screen.

Filling the Cyanide Leaching Vats.—From the screens, the dry dust falls into a long, narrow trough running parallel with the stamp-motors, along which it is conveyed to the dust-bin at one end of the mill by means of an Archimedean screw. From the dust-bin the pulverized ore is lifted by a bucket-belt elevator and discharged on to an 8 in. rubber-belt provided with rope edges, and by this conveyed to, and across, the dust-hopper, which is 110 ft. long, running the entire length of the cyanide-plant house.

The dust-hopper has twenty doors for discharging the dust into the trucks, which are run straight out over the leaching vats on to travellers running on rails. The travellers are provided with hand traversing gearing, thus enabling a truck to be tipped at any part of the vat. This is an important feature, as the finely pulverized material has a tendency to pack if moved about or touched in any way after being tipped into the vat.

As a further preventative against packing, there is a small traveller fixed below the main traveller, provided with a platform at the height the ore has to be filled up to. All the trucks are tipped over this platform, which breaks the fall of the dust, and throws it in a light shower all around.

The Cyanide Treatment.—The following particulars of the cyanide treatment were kindly supplied to me by Mr. E. G. Banks, the chemist of the cyanide operations.

The plant consisted of thirty-eight circular leaching vats, each 22½ ft. in diameter and 4 ft. deep, together with the necessary



IHI, N.Z.



dissolving and solution vats, sumps, extractors, vacuum-cylinders, solution and air-pumps, etc. :—

	Hours.
Filling vat, 30 tons, two men, ...	2½
Strong solution, 10 tons, 0.35% KCy leaching, . .	30
Weak solution, 7 tons, 0.1% KCy, with vacuum, .	15
First water wash, 6 tons, with vacuum, ...	24
Second " " " " " " " " " " " " " " " "	36
Discharging vat, one man sluicing, ...	2
Taking up and cleaning filter-bottom, ...	4½
Total, ...	114

The old vat-house contains ten rectangular concrete vats, five on each side. Each vat is 50 ft. by 40 ft. and 4 ft. deep.

Slime Treatment.—The slimes from the spitzluten and the overflow from the sand vats are mixed with lime-water (about 2 to 4 lbs. of lime per ton of slime) and conveyed to a nest of 36 V-shaped thickening boxes in which the slimes quickly settle, and are then drawn off in a fairly thick state from the bottom of the boxes. The overflowing clear water is returned to the stamps.

The thickened slimes are collected in six steel vats 32 ft. in diameter by 14 ft. deep. In these the slimes rapidly settle, and as the clear water overflows, it is stored for re-use in the mill and spitzluten.

When the vat is filled with slime-pulp to within 2 or 3 ft. from the top, the inflowing slimes are cut off and the charge allowed to settle for about 24 hours. Whatever clear water may be on top is syphoned off, and the thickened pulp (now about 1 of slime to 1 of water) is ready for treatment in the agitators.

The agitators are built of steel, 20 ft. in diameter by 6 ft. deep, and hold about 25 tons of slime (dry weight), together with about 40 tons of cyanide solution. The strength of the solution is about 0.1 per cent. The pulp is agitated by paddles, secured to a vertical shaft, actuated by overhead worm-gearing. The speed is 8 revolutions per minute.

After 24 hours' agitation the pulp is run to a pressure tank, and from thence is forced by compressed air into Johnson 6-ton filter-presses, where the gold bearing cyanide solution is extracted.

The zinc method of precipitation is used and does very good work, even on very dilute solutions.

Vanners are being erected to concentrate the more heavily mineralized ore. The tailings from the vanners will be mixed with the lightly mineralized ore-pulp, and pass through the course of treatment just described.

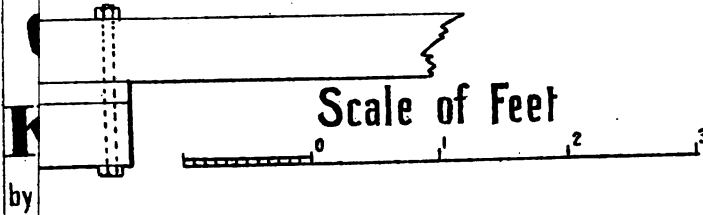
An extraction of 85 per cent. to 90 per cent. of the ore value can be recovered by this process, but no particulars of the actual recovery or costs are yet available.

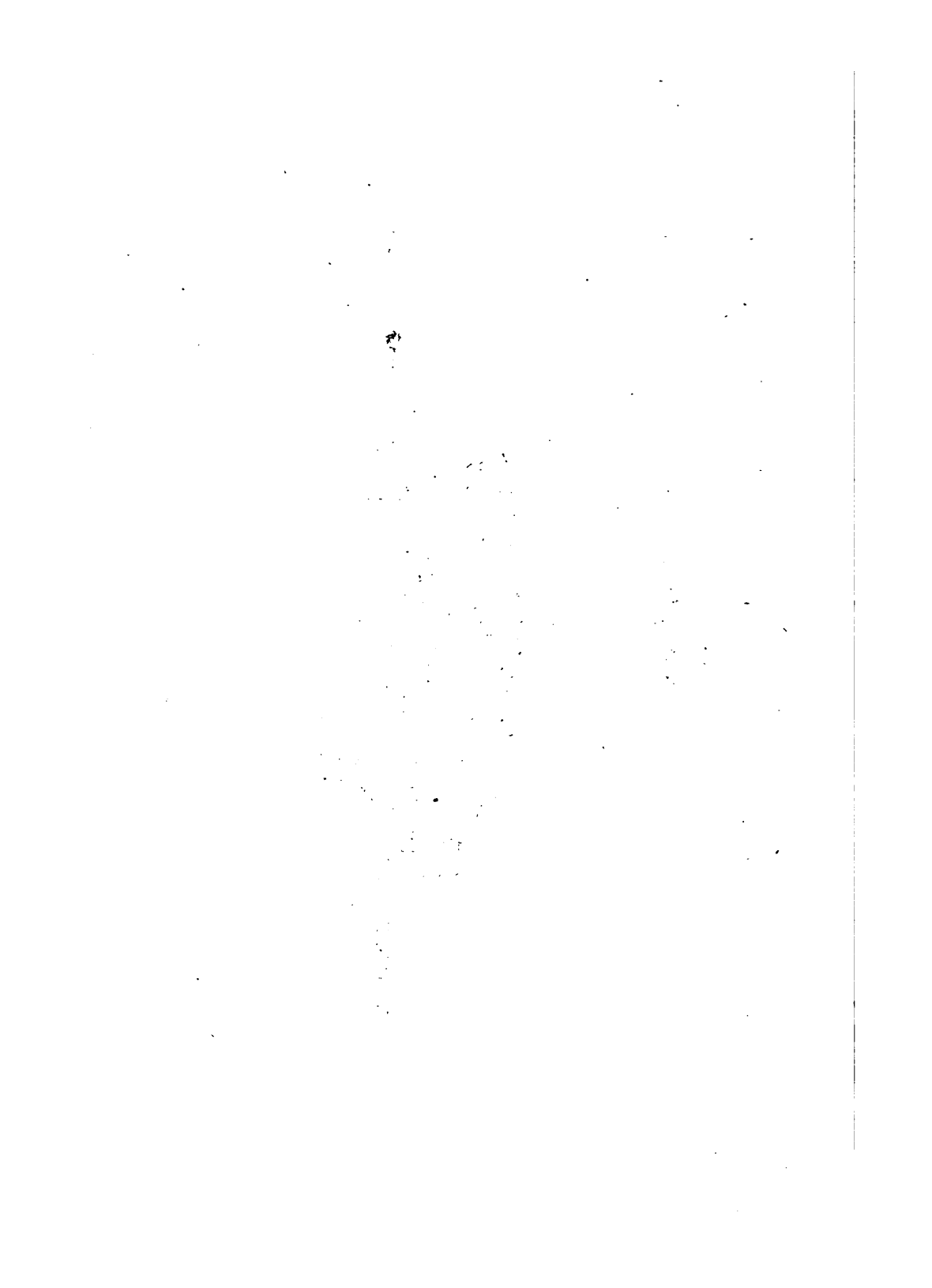
In the month of January of this year, the Waihi Company crushed 12,968 tons of ore, which yielded by cyanide treatment bullion valued at £29,055.

Waihi Union Mill and Cyanide Plant.—This mill consists of 40-stamps for wet-crushing and plate-amalgamation; sixteen circular steel vats for sand treatment (each vat having a capacity of 22 tons of sand); three slime-collecting vats, 32 ft. in diameter by 14 ft. deep; two agitators, 20 ft. in diameter by 6 ft. deep; and one Johnson filter-press, with a capacity of 6 tons of slime-dry weight, per charge. In addition there are the usual ore-

SECTIONAL ELEVATION

ACTATOR ARMS





breakers, spitzluten, pumps, air compressors, etc. The screens used are 40-mesh wire-wove steel, and the pulp after passing the amalgamated tables is elevated by a wheel-elevator to a height of 25 ft. and is here classified into sands and slimes by four spitzluten.

Sand Treatment.—The sands flow to the vats and are distributed by means of an automatic revolving distributor. A weak sump-wash, containing about 0·07 per cent. to 0·1 per cent. of cyanide, is run on, and this is followed by a solution 0·4 to 0·6 strong. The total time of treatment is from five to six days. The tailings are sluiced away through a bottom central discharge-door.

Slime Treatment.—The slimes, after the addition of lime-water (about 3 lbs. of lime per ton of ore), flow to the large collecting vats and are there thickened by natural settlement to about one part of slime to one of water. In these vats, a syphon provided with a ball-and-socket joint drains off the top clear water automatically. The clear water overflowing from the settling vats is sent to a reservoir to be pumped up for re-use in the mortar-boxes. This course is found to be necessary owing to the fact that even when the water appears to be perfectly clear it contains from $\frac{1}{2}$ grain to $2\frac{1}{2}$ grains of gold per ton, and as several tons of water are used per ton of ore, it is evident the loss would be from 6d. or 8d. up to two or three shillings per ton of ore, if the water were allowed to run to waste.

From the collecting or settling vats, the thickened slimes are drawn off to the agitators, situated at a lower level, in charges equal to 20 to 25 tons of dry slime, and are agitated for 24 hours with $1\frac{1}{2}$ tons of 0·1 per cent. cyanide solution to 1 ton of dry slime.

Waihi Mill.—This is the oldest of the Waihi Company's mills. Here the ore for some years was dry-crushed and pan-amalgamated by the Washoe process ; and afterwards dry-crushed, and the dry pulp cyanided by the direct process in the manner described by Mr. Barry in the preceding pages. Of the present treatment Mr. Banks writes in April of 1903 as follows :—

“ At the old 90-stamp mill dry-crushing was stopped at the end of 1902 and alterations necessary for wet-crushing rapidly completed. By the middle of January 1903, the mill was restarted. The ore (about 220 tons per day) is stamped through 40-mesh wire-wove steel screens ; passed over amalgamated tables, of which there are fifteen, each 12 ft. by 6 ft. ; and then over thirty union vanners which extract about $2\frac{1}{2}$ to 4 per cent. of concentrates. After passing the vanners, the pulp is elevated by wheel elevators to a series of sand separating boxes. Here the sands and slimes are separated, care being taken to keep the sand as free from slime as possible.

“ *Treatment of the Sand.*—This is effected in circular vats $4\frac{1}{2}$ ft.

deep and capable of dealing with 40 tons of sand per charge. The vat is first filled with water and the sand is then run in through an automatic revolving distributor. The overflow water (carrying a little fine sand and slime) runs into an annular launder and is conveyed back to the elevating wheels and joins the pulp flowing to the separating boxes. When sufficient sand has been run into the vat a 2 in. draw-off pipe, passing through the side of the vat, is lowered and the water drawn off. This water also contains a little slime and flows to the elevating wheels.

"After the charge has drained for several hours, a weak sump wash (testing about 0.08 per cent. KCy) is run on, and this is followed by a strong solution testing 0.5 per cent. KCy; when this solution has reached the bottom of the charge, percolation is stopped and the solution circulated by means of an air-lift inside the vat. After five days the strong solution is drawn off and the charge washed with weak solution and water. The tailings are then sluiced away.

"Treatment of the Slime.—After separation from the sand, the slime pulp (about 20 of water to 1 of slime) is mixed with lime-water and conducted over a nest of spitzkasten, thickening the slime-pulp to about 10 per cent. of slime, the balance of the water passing over in a clear state to be re-used in the mortars.

"The slime-pulp is now collected in one of two collecting vats, which are 14 ft. deep by 32 ft. in diameter, and fitted with an overflow launder. The pulp is run to the centre of the vat and discharged through a vertical box chute extending several feet below the top of the vat, so as to cause as little disturbance as possible. The pulp runs in until the slimes show at the top, the clear water overflowing to a reservoir for re-use. The collecting vats thicken the pulp to about 2½ of water to 1 of slime.

"The thickened slime pulp is now still further separated from the contained water by means of filter-presses, of which two are required for 'drying' the slimes, each dealing with about 40 tons (dry weight) of slime per day.

"The slime-cakes, now containing only 25 per cent. to 35 per cent. of moisture, drop from the presses on to a screen-conveyor which discharges into a disintegrator. This disintegrator is 7 ft. in diameter by 14 ft. deep and fitted with three sets of revolving arms bolted to a centre shaft driven by overhead gearing at the rate of 20 revolutions per minute. Cyanide solution (0.12 per cent.) flows in at the bottom of the disintegrator in such proportion that the overflowing pulp contains 1 of slime to 1½ of solution. This pulp flows through a series of four agitators each 14 ft. deep by 20 ft. in diameter, fitted with stirring gear making three revolutions per minute.

"The stirring gear consists of two arms secured one foot from the bottom of the vat to a central shaft driven by overhead worm-gearing. Effective agitation is produced by blowing compressed air through a number of pipes dipping down close to the stirring arms. The revolving arms serve to bring the pulp in contact with the escaping air, at which point fairly violent agitation is produced.

"The pulp enters No. 1 agitator at the top, flows from the bottom of No. 1 through a connecting pipe terminating half way up the side of No. 2. Similarly it passes from No. 2 to No. 3, and from No. 3 to No. 4. From No. 4, the pulp flows to a larger agitator, No. 5, which is 32 ft. in diameter by 14 ft. deep, and also fitted with stirring gear and air-agitation. The compressed air in the pressure tank after filling a filter-press is utilized by being blown through the agitators.

"The pulp is drawn off from No. 5 agitator, from time to time as required, to fill one of the four filter-presses used to complete the treatment. A press charge contains about $5\frac{1}{2}$ tons (dry weight) of slime. The cakes are 3 in. thick.

	Hours.
Time required to fill press at 60 lbs. pressure,	1
„ to wash at 75 lbs. pressure,	3
„ to open, discharge, and close press,	$1\frac{1}{2}$
Total,	$5\frac{1}{2}$

"From five to six hours can be taken as the time required to treat a press charge of Waihi slime. Of course, the more fine sand contained in the slime, the quicker a press will fill and wash. About 25 dry tons of Waihi slime can be taken as the average amount treated per press per day. The labour required is one man to each press each shift of eight hours. The cost for cloths, rubber-rings, etc., is from 2d. to 4d. per ton of slime treated.

"*Extraction.*—By amalgamation and concentration from 40 per cent. to 50 per cent. of the value is recovered, each process extracting about half that percentage. The sand treatment extracts about 80 per cent. of the gold and 60 to 70 per cent. of the silver contained in the sands. There is obtained by the slime treatment a recovery of 95 to 96 per cent. of the gold, and about 70 per cent. of the silver. The total extraction is about 89 per cent. to 90 per cent. of the gold and 75 per cent. of the silver.

"The consumption of cyanide per ton of ore varies from $2\frac{1}{4}$ lbs. on lightly mineralized ore, to 4 lbs. on heavy sulphide ore.

"The cost of the slime treatment is not given, but it will probably work out to about 6s. 6d. per ton of slimes treated.

"The output of the Waihi Company is about 18,000 long tons every four weeks of twenty-four working days, for a return varying from £50,000 to £52,000. The ore is pulverized in the company's mills, comprising 320 stamps, and then subjected to cyanide treatment by the various processes described above. The cyanide plants have a capacity of about 1000 tons per day."

From the foregoing description it will be seen that the slime process in use at the Waihi and Waihi Union mills is practically the same as that in use at Kalgoorlie in Western Australia. On the other hand, the process at the Waihi Company's Waikino mills is almost identical with that in use at Glencairn Main Reef, Johannesburg, differing only in the final stages in the adoption of filter-presses to separate solutions from the slimes instead of decantation which is almost universal in South Africa.

Waihi Tailings.—The treatment of the Waihi Company's tailings is very instructive. The tailings are the residues resulting from the pan-amalgamation of the dry-crushed ore before the introduction of the cyanide process. The ore was crushed through a 60-mesh screen, and pan-amalgamated in charges. The residues were discharged from the settlers into large dams, where they were allowed to settle. They mostly consisted of fine sands and a good deal of slimes. They contained no base metallic impurities, and the gold existed principally in the form of amalgam.

Some 25,000 tons of these tailings were successfully treated by the Cassel Gold Extracting Company, whose works have recently been acquired by the Waihi Gold and Silver Mining Company, who treated the remainder of the tailings on their own account. The plant consists of eight leaching vats, each 22½ ft. in diameter and 4 ft. deep, together with all the necessary appliances.

The details of the cyanide treatment adopted for the treatment of these tailings are given below in tabulated form:—

CYANIDE TREATMENT OF WAIHI TAILINGS.

	Hours.
Filling leaching vat, 30 tons, three men, ...	8
Preliminary lime or water-wash, 6 tons, with vacuum, ...	6

LEACHING—

Strong solution, 8 tons, 0·6% KCy, ...	30
Weak solution, 4 tons, 0·2% KCy (from strong sump), ...	12

WASHING, USING VACUUM—

First weak cyanide wash (from weak sump), 4 tons,	. 12
Second " " " "	. 12
Third " " " "	. 12
Fourth, water wash, 4 tons,	. 12
Discharging vat, one man sluicing,	. 4
Total,	. 108

REMARKS.—The tailings were generally very clean, and the preliminary lime or water wash was not always applied. The strong solution was allowed to stand in contact with the tailings about four hours before the percolation was commenced. The average value of the tailings was about 24s. per ton, and the actual extraction about 75 per cent., at the cost of 8s. per ton.

Try Fluke Mine.—At the works of this Company, at Kuaotunu, the ore was wet-crushed through a 40-mesh screen, and passed over amalgamated copper-plates. The tailings were run directly from the plates into settling-pits. When the pits were full, the slimes were removed from the lower end, and spread out to dry in the sun. When dry, they were broken up, and then were filled into the leaching vats together with sand, in the proportion of one truck of slimes to two trucks of sand.

The sand and slimes were thoroughly mixed in the vat before the solution was put on. The Company's metallurgist informs the author that the average time of treatment was as follows:—

	Hours.
Filling vats, 20 tons,	6 to 8
Strong solution, 5 tons, 0·6% KCy, standing	
in contact with tailings,	8 to 12
Percolating,	24 to 30
Weak solution, 5 tons, 0·2% KCy,	5 to 6
Weak cyanide washes, five of 5 tons each,	
0·1% of KCy,	25 to 30
Totals,	68 to 86

The ore consisted of grey and yellowish-brown quartz, sometimes containing a considerable proportion of iron and manganese oxides, the latter generally predominating. The greater part of the gold was excessively fine, being locally known as "float gold."

The average value of the tailings was about 20s. per ton; and the actual extraction by cyanide about 75 per cent., at a cost of 7s. 6d. per ton. When the tailings were of higher value than usual, they were turned over in the vat after the last washing and

washed again. In this case it was found that the extra extraction more than paid for the extra labour.

At the Kapai-Vermont mine, which adjoins the Try Fluke, the same ore was dry-crushed in a ball-mill, and then subjected to direct cyanide treatment with the most satisfactory results, the actual extraction generally exceeding 85 per cent. of the assay value. In this mine, shoots of very rich ore have been frequently met with, containing a considerable proportion of comparatively coarse gold. With such ores the strong cyanide solution was circulated through the leaching vats until an adequate extraction was obtained.

Waitekauri Mine.—At the Golden Cross section of the Waitekauri Gold Mining Company's Special Claim, near Waihi, the ore, before the introduction of wet-crushing at the new mill, was dry-crushed with stamps to pass through a 40-mesh screen, and treated directly with cyanide. As a small proportion of the gold was coarse, the tailings were passed over amalgamated copper-plates, 30 ft. long and 3 ft. wide, set with a fall of 1 in 12.

The details of the cyanide treatment of this ore may prove useful, and are given in the following tabulated statement:—

	Hours.
Filling vats, three men, 22 tons, ...	3½
Strong solution, 9 tons, 0·45% KCy, ...	48
Weak solution, 9 tons, 0·2% „ „	18
First weak cyanide wash, 5·5 tons, 0·05 to 0·15%, ...	18
Second „ „ 5·5 „ „	18
Third „ „ 5·5 „ „	18
Fourth, water wash, 3·0 „ „	18
Discharging vat, one man sluicing, ...	14½
Total, ...	156

The average value of the ore treated during 1894 was £4, 15s. per ton; and the actual extraction varied from 91 per cent. to 93 per cent., at a cost of 8s. 6d. per ton.

WESTERN AUSTRALIA.

TREATMENT OF SULPHO-TELLURIDE ORES.

At Kalgoorlie, the chief mining centre in the State, the gold occurs in the oxidized surface ores in a free state, and in the unoxidized ores in combination with tellurides and sulphides. In all cases it exists in an extremely fine condition in talcose

salcareous ores that possess a great propensity to form slimes. The successful treatment of these ores at first presented a difficult problem to metallurgists, and after repeated failures along old lines, a process was developed that in some respects possesses some peculiar features. And the early difficulties were not confined to the mere treatment of the rebellious ore. The climate was tropical and dry, water scarce and often brackish, and the distance from the seaboard great, and over arid plains. But these difficulties have now been overcome. A State railway connects the distant mining centres with the capital, and a State water supply, the greatest undertaking of the kind in the world, provides an abundance of pure water, carried in a pipe-line for a distance exceeding 300 miles.

The telluride ores are very brittle, and consequently a larger proportion of the gold goes into the slimes than into the sands. This circumstance and the slimy character of the ore led to the adoption of the cyanide process and filter-press treatment of the slimes. The object at many of the mills is to slime as high a proportion of the ore as possible.

The Diehl and Riecken processes have been installed at several mines with, it is said, satisfactory results. A detailed description of them will be found in Chapter XIII.

In some of the mills the slimes are drained or partially dried in presses, disintegrated and agitated with cyanide solution, and afterwards pressed and washed in presses. In other mills, the dissolution of the gold is effected directly in the presses.

I am indebted to Mr. F. B. Allen, M.A., B.Sc., Director of the Kalgoorlie School of Mines, for the following interesting details of the general methods of treatment adopted at that mining centre:—

General.—The essential features of the treatment are dry-crushing, roasting or not according to circumstances, fine sliming, leaching with cyanide, and filter-pressing.

The roasting is generally effected in Edwards or Brown straight-line furnaces. The former roast from 14 to 16 tons per day, at a cost of 4s. to 4s. 6d. per ton; and Brown, 30 tons per day to 0.1 per cent. sulphur as sulphide for 7s. to 9s. per ton.

Ore containing over 3 per cent. of moisture is dried before dry-crushing in ball-mills. No 5 Krupp, running at 25 revolutions per minute with 15 H.P., crushes 25 tons through a 40-mesh screen for 1s. 2d. per ton, and of this from 65 to 75 per cent. will pass a 100-mesh sieve. The Griffin mill, when crushing to a 15-mesh, will form a product of which 75 per cent. will pass through a 100-mesh sieve, at an approximate cost of 2s. per ton.

Great Boulder Proprietary.—The sulphide ore is partly

crushed wet with stamps and amalgamated in Wheeler pans, and partly dry-crushed in Griffin mills, roasted, fed into mixers which supply Wheeler pans with the pulp, which is further ground and amalgamated with mercury without the use of copper-plates.

Cyanide Process.—The pulp from the continuous overflow of the pans is led into settlers, first passing, in the case of sulphide ore, over canvas tables to eliminate concentrates. About one-third of the gold contents are recovered by amalgamation. The furnaces * used are Edwards (16 tons per day), fed with producer gas, and they roast the ore down to 0.11 per cent. sulphur as sulphide, and 2.0 per cent. sulphur as sulphate. Push conveyors are used throughout, and samples are taken automatically as the ore is discharged from the elevators.

The fine slimes from the settler go to a settling tank, whence they are lifted up and passed through spitzluten, the heavier particles being returned to the pans. The slimes, 1 to 1, are agitated and passed to montejus or pressure-tanks, and then to filter-presses.

The four Dehne presses hold 4 tons each; the five Martin presses $3\frac{1}{4}$ tons, forming 3 in. cakes. They are worked by hydraulic pressure.

The solutions are clarified by Excelsior presses, passed through three zinc boxes, and the gold-slimes treated with sulphuric acid and melted in tilting furnaces.

The residues are dumped by a Ledgerwood hoist on to a 60 ft. heap.

Kalgoorlie Mine.—Part of the ore is dried in a White-Howell drier, the drier ore passing direct to large 200-ton storage bins.

The ore is automatically fed into six No. 5 Krupp ball-mills, crushed through a 35-mesh screen, so that about half will pass through a 120-mesh, and then led into a 400-ton bin, whence by roll feeders it is fed into nine Edwards roasters, each treating 15 tons per 24 hours.

The roasted ore falls on to a push-conveyor and is carried to a bucket-elevator, which lifts it to a mixer. The pulp from this is separated by a series of conical spitzkasten into slimes, which pass on through several pyramidal spitzkasten to have the surplus water removed; while the sands and concentrates run over copper-plates 10 ft. long and Halley tables, from which the concentrates are ground in Wheeler pans. The sands are sent to three 100-ton vats, drained of water, and bottom-discharged into steel vats below. Ordinary cyanide treatment occupies three weeks, and the cyanide solutions as they come off are returned to

* Three-hearth Merton furnaces are now being used.

the top of the sands by an air-lift, which keeps the solutions constantly circulating.

The slimes from the spitzkasten thickened up to 1 in 1 flow to a set of five pressure tanks $5\frac{1}{2} \times 13\frac{1}{2}$ ft., where they are agitated with compressed air and cyanide solution for four hours. One tank is being filled while another is being emptied and the others working, the air passing from one to the other. The agitated pulp is then filter-pressed. Cyclone settling cones are used for settling the dust, as well as an air-lift for the pulp.

Golden Horseshoe.—The oxidized ore is screened through 2 in. grizzlies, the coarser lumps passed through No. 3 Gates crushers, and the broken ore stored in a 200-ton bin, whence it is trucked to Challenge ore feeders supplying a 50-stamp battery of 1000-lb. stamps. The ore is wet-crushed through a 24-mesh woven wire-screen at the rate of 4 or 5 tons per 24 hours, the greater part passing a 100-mesh screen. Free gold is amalgamated both inside the boxes and outside on copper-plates, which are followed by concentrators of the Wilfley type. The coarse sands are ground and amalgamated in a series of pans followed by settlers and spitzkasten. The fine sands thus obtained are lifted by a 42 ft. tailings-wheel to vats fitted with Butters distributors. The slimes pass over 30 ft. of canvas tables for the further elimination of fine concentrates. Double treatment of the sands, occupying nine or ten days, is practised. The fine concentrates, etc., are collected and sent to the smelter. The slimes are pumped direct, without agitators or montejus, to nine Dehne presses, each carrying fifty 3 in. cakes, pressed, washed with cyanide solution and water, and discharged. Excelsior presses are used for clarifying the gold solutions, which are then passed to four zinc extractor boxes.

Sulphuric acid treatment with filter-pressing is adopted for treatment of the zinc slimes, which are finally melted in a tilting furnace.

Sulphide Ore.—The new battery is of 50 stamps, each 1250 lbs. The ore, after passing the Gates crusher, is brought by a Robins belt to Challenge feeders and crushed wet through a 30-mesh screen, passed over copper-plates, and lifted by a tailings-wheel to a series of grading boxes supplying Wilfley tables. The slimes which overflow from the graders, together with that from the Wilfleys, are lifted again to three hydraulic classifiers, the slimes passing to settling tanks and the sands to five flint-mills which crush them to 220-mesh, and then are again elevated to the graders and finally passed to the settling vats. The thickened slime produced in these vats is run into steel agitators, agitated with cyanide for 24 hours, pressed in six 5-ton presses, and the

waste material dropped on to Robins belt conveyors for removal. Rich sulphide ore is sent to the smelters.

Great Boulder Main Reef.—The *Sulphide ore* is treated without drying by breaking in a No. 3 Gates crusher, delivering it by a Challenge feeder on to a Robins belt, and then to two Krupp ball-mills, where it is crushed to 30-mesh, a large portion, especially when the ore is schistose, passing 100-mesh.

The crushed ore is roasted in a Richards shaft-furnace 65 ft. high, with eleven floors, and rabbled by hand, especially on the lower floors. In this furnace 35 tons per day are treated. There are also three Edwards furnaces, each of a capacity of 12 or 15 tons, which are bricked up to a constant angle.

The hot roasted ore falls into a launder carrying dilute cyanide solution, and is raised by a tailings-wheel to a spitzkasten. The sands are ground fine in Wheeler pans containing mercury, but no plates, and with a continuous overflow into another spitzkasten, from which the sands again pass to the pans, while the slimes join the first slimes, which are pumped into agitation vats. These vats are 21 ft. in diameter and 6 ft. deep. The vats are allowed to fill in 7 hours, the solution gaining in cyanide during the last foot of filling from cake cyanide then added.

After 10 hours' agitation the pulp is discharged or run off into montejus and filter-pressed in 3 in. cakes (Dehne).

The solutions are clarified by passing through a small press, and led into zinc boxes. The gold slimes are washed, pressed, treated with sulphuric acid, dried in a large iron muffle, and smelted in Cornish furnaces.

	Hours.
Dehne: filling in, ...	$\frac{1}{2}$
Leaching and washing	$1\frac{1}{2}$
Discharging, ...	$\frac{1}{2}$
Four tons in, ...	$2\frac{1}{2}$

Lake View.—*Sulphides, roasted.*—From Gates No. 5 crusher the ore passes to 400-ton ore-bin, and thence by aerial tram to ball-mill bin. There are four Krupp mills with a capacity of 130 tons per day, 65 per cent. of which will pass a 150-mesh screen.

Ore is led by conveyors to four Brown straight-line furnaces 180 ft. $\times$ 10 ft., roasting 30 tons per day down to 0.2 per cent. of sulphur as sulphides. The roasted ore is elevated into 50-ton agitators into which 0.15 per cent. KCy is running, and the resulting sands are transferred into ten leaching vats, while the slimes pass on to three agitators for further agitation, before being passed through montejus and Dehne presses. Then follows zinc precipitation and sulphuric acid treatment.

After roasting, analysis shows sulphur as sulphates, 2·82 per cent. ; sulphur as sulphide, 0·32 ; sulphur volatilised, 0·46. Total sulphur, 3·60 per cent.

In a recent report on the Riecken process at the South Kalgoorlie mine, Mr. R. Hamilton gives the costs as follows :—

	s.	d.	
1. Coarse crushing and transport to mill, .	2	7	per ton.
2. Fine pulverising, including proportion of general charges and power, .	5	3	„
3. Roasting and conveying ore, including pro- portion of general charges and power, .	7	0½	„
4. Agitation with cyanide, and electrical pre- cipitation treatment, .	8	5½	„
5. Filter-pressing and washing, .	6	3	„
Total charges, .	£1	9	7½ „

The average value of the ore for January, February, and March was £3, 17s. 6d. per ton, and the value of residues 2·125 dwt. of gold, equal to a value of 8s. 10½d. per ton, representing an extraction of 88·4 per cent.

COSTS, FEBRUARY 1903.

Ivanhoe Mine.

Probably lowest costs on the field :—

	s.	d.
Mining, ...	13	10·73
Reduction, ...	11	9·65
Development, ...	7	11·85
Capital, ...	2	3·03
General, ...	3	0·55

38 11·81 per ton.

11·098 tons for 10,318 ozs. of gold, valued at £43,800.

Lake View Mine.

	s.	d.
Stopping, ...	9	5·6
Diehl process, ...	17	11
General, ...	3	6·3

30 10·9 working costs.

Mine development, ...	13·9 approx.
Additions to plant, ...	4·9 „

Kalgoorlie Mine, 46s. per ton (higher than usual).

NEW SOUTH WALES FILTER-PRESS PRACTICE.

The following interesting details are extracted from a paper recently written by Mr. I. W. Rock.* The plant to which these details refer is one capable of dealing with 200 tons of dry slimes per day. A summary of the process is as follows:—

1. The dry slimes are discharged into a mixer, in which they are ground up, while cyanide solution is added, the mixture out-flowing into a storage tank, whence it is elevated into agitator vats by means of a centrifugal pump.
2. Mixture agitated for a period ascertained by experiment, by some mechanical means, thus ensuring intimate contact between the metallic and chemical particles.
3. Contents run into montejus, which consist of cylindrical vessels provided with inlet and outlet valves, and also connections to the receiver of an air compressor.
4. Forcing the mixture by means of compressed air from the montejus into filter-presses, the gold-bearing solution escaping through the filter-cloths of the presses, while the solid material remains in the frames, forming large thin cakes.
5. A further extraction from these cakes by "washing" or forcing, at a high pressure, a weaker solution of cyanide through them, and, if found necessary, a second washing with water only.
6. Getting rid of the exhausted slimes by opening the presses and discharging the cakes into trucks, which convey them to a dump.
7. Clarifying the extracted gold solutions flowing from the presses, by running them through sand traps and thence into tanks, pumping them into elevated tanks, in order to obtain regular pressure, and passing them through a finer class of filter-presses, in order to retain any impalpable solid matter which would foul the zinc extractor boxes.
8. Passing the outflow from the presses into the extractor boxes and treating therein in the usual manner, the outflow being of course returned to sumps for strengthening and re-use.

Mixers.—These are made as a standard article by the engineering trade, and consist of a steel cylinder 6 feet in diameter by 5 feet deep, lined with an inserted cone, which has openings in it near the top of the outside shell, and at the bottom a four-bladed propeller on a vertical shaft, with mitre wheels and driving gear. The propeller not only cuts up the slimes and mixes them with the solution, but it drives the mixture upwards through the openings and up to the outlet, whence it overflows into the storage tank, on its way to the agitators.

Agitators.—These consist of steel vats 16 feet high and 8 feet diameter, open at the top, and with the lower portions tapered to openings which connect with both outlet and circulating pipes.

* Rock, *The Australian Mining Standard*, Dec. 12, 1901.

To each of these agitators there is connected a centrifugal pump, whose sole duty is to draw the contents of the vat from the bottom and redeliver it into the top. This mode of mixing the particles and bringing them into intimate contact has been found highly efficient and economical in regard to time as compared with arms or paddles adopted in horizontal agitators previously adopted.

Centrifugal Pumps.—These circulating pumps, and the one for elevating the mixture into the agitators, are the only ones used for dealing with gritty material in the process. They have the usual water-pressure chambers in connection with the glands, but as water would dilute the mixture and upset the proportion, they are fed with cyanide solution under pressure from an elevated tank, supplied by a small pump.

Montejus.—These are each large enough to receive the contents of an agitator, made after the style of a compressed-air receiver, buried vertically in the ground. Inlet and outlet valves, of chemical and grit-proof make, are fitted on them; compressed-air supply and exhaust, test cock and pressure gauge, are fitted on each. In addition, there is a small air-agitation connection, so that any mixture remaining for a time in one is kept alive and the deposition of solid matter prevented.

Filter-Presses.—These are generally made by or after the pattern of Dehne, the German engineer, the usual size having fifty 40-inch $\times$ 40-inch $\times$ 2-inch frames, the total contents being 75 cubic feet. In addition to the fittings supplied by makers, there are several others which are necessary and have to be added, such as drip trays, slimes and leak launders, and several other minor accessories. Of course the discharge shoots for delivering the cakes of spent slimes, as they fall out of the frames into the trucks beneath, form part of the setting of the presses in the building prepared for them.

Wash Solution Pump.—This may be of any description of high-pressure pump with cast-iron fittings. The pressure for filling the presses from the montejus may be reckoned at up to 80 lbs., that of the wash-water at 100 lbs. to the square inch. It may be noted here that both after filling the presses and after washing the contents, compressed air is admitted for a short period to drive out any moisture.

Clarifying Presses.—These are also filter-presses, but of a much lighter description, the frames being usually made of wood. They are supposed to be self-cleansing, by reversal of the flow through them, and are so with some materials, but other slimes are so gluey that it is found necessary to have a spare set of filter-cloth frames, which can be substituted for the foul ones in

a few minutes, the latter being taken away and scrubbed. These presses are very delicate, and the pulsation of a pump destroys their efficiency. It is therefore necessary to pump the gold solution into a tank about 18 feet high, whence it flows quietly through the presses, and thence to the extractor boxes.

The items of storage vats, sumps, etc., need not be particularly specified, and it need hardly be added that the adjuncts of sufficient steam power, compressed air, and supply of water are necessary.

Practical Work.—If a plant as above described is doing its ordinary work, the routine would be as further described. Side-tipping trucks are delivered at regular intervals alongside the mixer, and the contents dipped bodily out, the fall being broken by a shoot, sufficiently flat to require the attendant to use a shovel and in some degree regulate the feed. He also, through practice, knows how much cyanide solution to run in from a supply cock, so that a fairly constant flow of mixture escapes into the storage tank.

When one of the agitators is empty, a signal is given to refill, and the elevated centrifugal does this in a few minutes from the storage tank, the circulating pump running all the time. After sufficient agitation, the man in charge of the machinery room opens the inlet valve and fills one of the montejus, the air escape cock being open. When the charge is all run in he closes the inlet valve and opens the little agitation cock until he receives a signal that a press is to be filled. When he gets that he closes the above, opens the outlet valve, and then the compressed air cock; the pressure gauge rises, and in a few minutes half the contents of the montejus has been transferred to the press in the room above. In this room the signal to fill was given when any one of the presses had been emptied and reclosed, ready for filling. All the outlet cocks on the frames are open, and the gold solution at once flows from them until the press is full of solid matter. Notice is then given to close the valves previously open and start the wash water pump; the taps are closed, the wash water outlet opened, the flow therefrom directed for a little time into the same launder as the gold solution ran into, and thereafter into a main leading to a sump for re-use. Pumping is then stopped, a little air blown through to complete the process, and the press unscrewed, the drip trays being previously removed. One after another the dummies, as the slimes-bearing frames are called, are pulled forward, the cakes either fall or are pushed out into the trucks below, each frame is scraped clean with a scalpel, and the press tightened up again for a fresh charge.

The treatment of the gold solution is pretty well automatic; its

flow through the clarifying presses has been already described, and its final delivery into the zinc boxes of the extractor-house. It will be understood that the diagram in no way indicates any arrangement of the plant. Accessibility and ample space are particularly necessary, and some details of the plant which are purchased as articles ready for use from the makers require considerable alteration to make them suit the above requirements.

Other States in Australia.—In Victoria, South Australia, and Queensland, the cyanide process is employed almost exclusively for the treatment of sands and old accumulations of tailings. Nowhere has any distinctive feature been developed except in Victoria, where charcoal is much used for precipitation purposes instead of zinc. This is really a survival and adaptation of the charcoal precipitation process formerly employed in Victoria in the chlorination process.

UNITED STATES.

The cyanide process has been successfully introduced in the states of California, Colorado, Idaho, Montana, Nevada, Utah, New Mexico, and Washington, and Black Hills district, South Dakota. Generally speaking, the adoption of the process in the States has been slow, the primary cause for this being doubtless due to the complex character of the ores. The progress of the process, however, during the past year or two has been very marked.

The process is one presenting many difficulties, especially with ores containing base sulphides, and up to the present time American metallurgists have been content to feel their way on safe ground.

The cyaniding of tailings is conducted on the same lines as elsewhere, while the slimes problem seems to have been successfully solved by the adoption of agitation followed by decantation.

The treatment of the sulpho-telluride ores of Cripple Creek districts by cyanide after roasting has been attended with much success. Up to the present time, the electrical precipitation of gold from cyanide solutions has had little or no application on a working scale. For the treatment of high-grade slimes, there may possibly be scope for the filter-press process practised in Western Australia.

The new Homestake plant, with a capacity of 1200 tons per day, and the Smuggler-Union of 600 tons per day, are among the largest in the States. Actual working details of cyanide treatment in the States are seldom available, and the following notes will be read with much interest.

Colorado.—*Camp Bird Mills, Ouray.*—I am indebted to Godfrey Doveton for the following instructive particulars of the cyanide treatment of tailings at this mill. The tailings contain a small percentage of copper, which has led to the adoption of some interesting modifications of the usual practice. Mr. Doveton says the treatment of the tailings presents no new feature, but as much of the gold is partially enclosed in the coarser particles of sand, a rather lengthy treatment is required.

The tanks are charged with Butters distributors, and the slimes overflow from three slime gates placed at the sides of the vats, and are conducted to the slime dam and settled. Slime gates are preferred to a circular launder, and as a somewhat better classification is effected, a better leaching is the result.

A sizing test of a large number of vat samples showed the following results:—

Retained on 40 mesh, ...	5	per cent.
60 "	20	"
80 "	13·13	"
100 "	15·00	"
120 "	12·80	"
150 "	17·50	"
Passed 150 "	15·62	"
Total	99·05	

The ore was crushed in stamper batteries through a 35-mesh wire-wove screen.

On assay, the material found on the 40 and 60 meshes was found to run considerably higher than the finer material, and the bullion button was very much finer than that resulting from the assay of the finer product.

The vats are sampled at the distributor nozzles as a check upon the vanner tailings at the stamp mill. When filled the charge is also sampled with a borer, some 40 to 50 bores being taken from a 500-ton vat. An acidity test for free and combined acid is made on the head sample, and the requisite quantity of lime is found, and added on top of the charge, and well mixed by shovelling over the top layer of material.

Details of Treatment of 100-ton Charge.—The tank being partially drained and lime added, 20 tons of a weak solution are run on, containing about 0·05 per cent. KCy. This solution, which also contains a considerable percentage of copper cyanide, acts as a food for the cyanicides, and also, owing to the presence of much cuprocyanide, dissolves much of the copper contained in the charge, thereby leaving the material in a better condition for the actual

working solution. No gold is dissolved as a rule by the preliminary weak solution, and as the whole of the available cyanide is destroyed, it is allowed to drain off and run down the sluice launders, to be saved for sluicing purposes, etc.

It should be mentioned that on rare occasions, when a portion of the gold contents is amenable to treatment, from 10 to 25 cents value of gold is removed by the first solution.

The weak solution is immediately followed by a succession of 10-ton charges of solution of 0.25 per cent. free KCy, until the outgoing solution rises to about 0.20–0.225 per cent. KCy. Usually 60 tons of solution are run on, and by the time the outgoing solution is up to near standard strength, 65 to 70 per cent. of the gold is dissolved. The charge is now allowed to macerate from 8 to 16 hours, and is subsequently rapidly drained, sampled, and shovelled to the tank below for the second treatment. During the shovelling the charge sample has been assayed, and the gold yet capable of extraction ascertained. Usually almost all the soluble AuKCy_2 compound has been carried out by the charges of solution. Should the assay show that much of the previously insoluble gold still remains in the ore, standard solution is added in the lower vat. Should the gold contents be unusually high, 0.3, 0.35, or 0.4 per cent. solution is used for the saturation of the shovelled charge. If, however, a fairly good extraction has been obtained in the upper vat, 0.25 per cent. solution is used, and when we are crowded with tailings weak washes are applied immediately after the saturation.

Mr. Doveton continues:—"Our usual practice is, however, to saturate with 0.3 per cent. solution, and allow to macerate from 4 to 6 hours, then drain rapidly, and apply a couple of washes of 0.3 per cent. solution of 10 tons each, drain partially dry, and give 25 tons of weak solution (0.08–0.125 per cent.). Before the last of the weak washes has disappeared below the surface, the charge is allowed to macerate two hours. This ensures all of the dissolved gold being carried out.

"The charge is now drained dry on surface, and water washes added till cyanide values fall to 0.04 per cent. The liquor is now transferred from the weak gold tank to the waste gold tank, and the solution, low in cyanide, is passed through the waste zinc boxes, and thence out of the mill. The gold value of the solution at the time of transfer from weak to waste zinc boxes is about 40 cents per ton, and much of this is precipitated in the waste zinc boxes.

"At the time of sluicing, the charge is fairly dry, containing about 12 per cent. of moisture, and the outgoing solution usually contains about 0.02 per cent. of free KCy, but only a

little double cyanide. As is usually the case, the first weak solution flowing from the charge is high in double cyanide, which we are now about to recover by our recovery process.

"It is noticeable that but little gold is carried out by the solution till the outgoing strength is almost up to standard strength of cyanide.

"A series of solution assays were made on a number of vats during treatment. Samples of the outgoing solution were taken hourly, and the samples for each six hours assayed together. It was found that up to 0.05 per cent. KCy the solution carried scarcely any gold at all. Between 0.05 and 0.1 per cent. the value increased from 40 cents to 150 cents per ton; from 0.1 per cent. to 0.15 per cent KCy it was about 225 cents; and at 0.2 to 0.25 per cent. KCy the value remained about 5 dollars per ton of solution. The value seems to remain pretty constant till the solution falls to 0.2 per cent KCy, when a gradual decrease sets in, and at 0.05 per cent. KCy the value is usually 80 cents per ton, and at 0.02 per cent KCy the value is seldom more than 25 cents.

"There is a considerable quantity of copper in the solution, but nevertheless we have a most perfect precipitation. Our aim is always to precipitate the copper on the zinc with the bullion, thus keeping the copper contents of the solution at a constant figure.

"Our strong sump assays for over a year have seldom gone as high as 40 cents, the average being from 15 to 20 cents. Our weak sump assays show but an average of usually from $2\frac{1}{2}$ to 4 grains of gold per ton by the assay of very large samples, say 50 assay-tons.

"The precipitation is usually practically perfect the week following the clean-up. With our waste zinc boxes we are not quite so successful. The copper contents of the incoming solution are usually high, and the gold value only some 25 to 45 cents per ton. The available cyanide is very low, varying from a trace to .025 per cent., with the total cyanides 0.05 per cent.

"By coating the zinc with mercury by immersing it in a weak solution of mercuric cyanide—made from mercuric chloride and KCy—we are enabled to make a very fair precipitation.

"When the weak solution runs rather lower than usual in gold value, more of the copper is precipitated on the zinc, and a somewhat less perfect precipitation of the gold takes place. The addition of concentrated cyanide solution at the head of the bin does not seem to help the precipitation of the bullion at all, but greatly accelerates the plating of the copper. The practice of strengthening the solutions entering the zinc boxes has been entirely discarded; it has been found here most advantageous to

promote the precipitation of the copper along with the bullion, thus preventing the solution becoming overcharged with copper. Occasionally, after placing fresh zinc in the last compartments of the boxes, it will be perfectly coated with copper in about eight hours, because the solution, when it reaches the last compartment or two, is very low in bullion value. However, in the course of 24 hours, when perhaps a higher bullion value is entering the boxes, the gold will be plated over the copper, the precipitate being a very lustrous black.

"Should the coppery zinc show no signs of precipitating bullion, precipitation is readily aided by the use of a few Winchester quarts of strong mercuric cyanide solution, which is allowed to drip into the compartment affected. When the zinc becomes gray in colour from the amalgamation of the deposited mercury with the coppery zinc, the addition of mercuric solution is discontinued, and in a few hours the zinc will become black from bullion precipitate.

"The evolution of hydrogen is much increased, should the solutions contain more copper than usual, and on several occasions, when the solution entering the boxes carried as much as 0.045 per cent. Cu, and the usual bullion contents, 4 or 5 dollars per ton, the evolution of hydrogen was so great as to lift the zinc partially out of the boxes; however, a good bullion precipitation was obtained, the solution leaving the boxes worth only 5 grains of gold per ton. An example of this was noticed recently, when a complete analysis of a working solution showed the presence of a small quantity of manganese, about 0.00778 per cent., estimated on the evaporation of 2 litres of solution by Volhardt's method. No sulphates or magnesia were found, but lime was present to the extent of 0.01387 per cent. of Ca. There was found a little lead, considerable alumina, no ferrocyanide, but a large amount of sulphocyanide, of the latter about 0.1078 per cent. There was a little iron present, but in what combination could not be determined.

"No alkaline sulphide has yet been detected in the solutions here, and it would seem that when zinc is used as the precipitating agent no alkaline sulphide can exist, the zinc being readily precipitated as sulphide from the ZnK_2Cy_4 .

"The oxygen is frequently estimated in the solution by a modification of Thresh's iodimetric method. The estimation is very valuable, as it enables us to ascertain whether our solutions are sufficiently aerated or not. Numerous oxygen tests showed the presence of from 2 to 7.36 mgrms. of oxygen per litre of solution.

"Experiments and numerous estimations were made to test the merit of employing jets of compressed air in the sumps and

storages, should it be found that the solutions received insufficient oxygen during their passage from the end of the zinc boxes to the leaching vats. However, we found that in the majority of the cases the solutions were capable of dissolving but a very little more oxygen on aeration, and the greater portion of that dissolved was diffused again on standing for any length of time.

"The weak solution was found, on the average, to contain a little more dissolved oxygen than the strong, the increase being most noticeable when the weak solution leaves the zinc boxes. Here the weak solution containing 0.08 per cent. KCy and 0.11 per cent. total KCy contained 0.64 mgrm. per litre of dissolved oxygen, while the strong solution contained only 0.4 mgrm."

*Cost of Cyanide Treatment at Camp Bird Mills.
5000 tons Tailings monthly.*

	Cents per ton.
Cyanide, 1 lb. per ton, ...	27.00
Zinc, 0.55 lb. ,, ...	5.00
Lime, 1.8 lb. ,, ...	1.30
Assays, ...	3.60
Labour, ...	18.15
Power—steam, ...	6.00
Sand-pumps, maintenance, ...	4.32
Electric light, ...	1.20
General repairs, ...	0.40
Supervision, etc., ...	4.60
Total, ...	71.57

Doveton gives the total charges as about 3s. per ton of ore treated, including 2.4 cents per ton for sulphuric acid treatment of gold slimes.

Smuggler-Union Mine.—The cyanide plant at this mine was erected last year. It has a capacity of 600 tons per day. Zinc precipitation is used.

Dorcas Pneumatic Cyanide Mill, Florence.—This plant has a capacity of 120 tons per day. The treatment here is of some interest, as compressed air is used to aerate and agitate the solutions and pulp in the leaching vats. The following description is extracted from a paper by Dr. Wells.*

The ore, averaging 20 dollars in value, passes through crushers and coarse rolls to the sampling room, thence through a revolving dryer and through two sets of finishing rolls which crush it to

* *The Engineering and Mining Journal*, Jan. 4, 1902.

24-mesh size. The finely crushed ore is then roasted in a Holt-hoff-Wethey furnace, the roasted product passing to the leaching tanks.

There are six of these tanks, 30 ft. in diameter, 4-5 ft. deep, and fitted with air pipes in the bottom for the introduction of air during the leaching. The air is supplied at a pressure not exceeding 5 lbs., about 1 cubic foot of air per minute to each ton of ore being sufficient for agitation and oxygenation.

The treatment generally lasts five days, and leaves a value of about 1 dollar per ton in the ore. The residue is then sluiced out and concentrated on Wilfley tables. The tailings after this treatment average about 40 cents per ton.

The tanks are filled by a conveyor, and as soon as the bottom of the tank is covered with pulp a solution containing 10 lbs. KCy per ton is run in gradually, the pulp at the same time continuing to flow in until the tank is full. The air is then turned on gradually, and is kept on until the pulp shows an extraction of at least 90 per cent. Whenever the air comes up unevenly through the charge the ore has to be stirred by men with iron rods. This generally requires about 30 hours. The air is then shut off and the pulp allowed to settle for one hour. Percolation is then begun and the strong solution run off as quickly as possible, followed by a weaker one of 5 lbs. KCy per ton. This operation is continued until the tests of the solution show only traces of gold. Water is then added to displace the KCy solution. The loss in cyanide is stated to be less than 1 lb. per ton. The dust resulting from the dry-crushing is collected and treated with cyanide, without previous roasting. From dust assaying 51.20 dollars per ton the tailings only contained 80 cents per ton. It is added to the roasted ore in the leaching tanks, 3 tons to each tank, spread evenly on top of the charge.

Cripple Creek Telluride Practice.—The country rock of this district is andesite breccia, phonolite, and decomposed granite. On the surface the ores are oxidized and carry iron peroxide, manganese oxide, and oxide of tellurium. Below water-level, the gold occurs in the minerals calaverite and sylvanite, and is always associated more or less with iron pyrites.

The gold in the surface ores is free, but not amenable to amalgamation, being coated with metallic oxides. It is, however, easily extracted by cyanide.

The unoxidized telluride ores have to be subjected to a preliminary dead roast before cyanide treatment.

The reduction of the ore is generally effected with Krom rolls, instead of the mills used at Kalgoorlie, the ore being crushed to pass through a 40-mesh screen. The roasted ore is leached by

percolation in vats with two solutions of cyanide. The stronger, containing from 0.5 to 0.75 per cent. of potassium cyanide, is allowed to percolate for 50 hours, after which the weak solution is added. The time of treatment varies from 70 to 100 hours.

Precipitation is effected with zinc.*

The treatment of the sulpho-telluride ores of Western Australia is obviously a more difficult problem than that of Cripple Creek ores. At Kalgoorlie, the ore is talcose and is highly calcareous, and forms so large a proportion of slime that ordinary percolation is impossible, and decantation too slow and costly. The successful solution of the difficulty was found in the use of filter-presses, in which the dissolution of the gold is effected under pressure, and by means of which the slimes are easily separated from the gold-containing solutions and washes.

As at Cripple Creek, the oxidized ores are treated in the raw state, while the undecomposed ores are subjected to a dead roast before cyanide treatment. With the adoption of filter-presses, it was soon recognized that the greater the proportion of slimes the better, and to attain that object, pulverizing mills, instead of rolls, have come into general use.

California.—*California King Mines.*—The plant at this mine has a capacity of 1000 tons per day. The ore is crushed by rolls to 20-mesh size and then treated in ten cyanide vats, each 40 ft. in diameter and 5 ft. deep, provided with bottom-discharge doors. Zinc precipitation is used.

There are many cyanide plants scattered throughout the State, including the large 140-stamp cyanide mill of the Golden Cross Mines at Hedges, but no particulars of these are available.

Montana.—Dr. Wells says that cyaniding has made great progress in this State. Many new plants have been erected, some of them of large size.

The process as carried on in this State is much the same as elsewhere. In many of the mills, the method is wet-crushing, amalgamation on plates, followed by cyaniding of tailings. In other cases the ore is crushed in rolls and cyanided direct. With slimy ores, the practice is to agitate and decant.

Nevada.—The sun-drying slime treatment adopted at the Dexter plant at Tuscarora has already been referred to in the chapter dealing with slime treatment. Of the other plants in operation in this State some are treating tailings, while others are dealing directly with dry-crushed ore.

New Mexico.—The Cochiti Company of Bland has a successful

* Prof. Furman, *Mines and Minerals*, January 1897.

method of treatment which possesses some novel features.* Here 98 per cent. of the dust will pass a 100-mesh screen. The process used is agitating and settling, the agitation being effected by steam and compressed air, the latter being under 60-lb. pressure. Steam is used to heat the charge, as it was found that the air on expanding cooled it too much. The mixture of air and steam is admitted through 1 in. horizontal iron pipes in the bottom of the tank. The pipes have 0.0625 in. holes in the under side for the escape of the air and steam. In 24 hours 94 per cent. of the total values are in solution, using a solution of cyanide of 3 lbs. per ton, but only 80 per cent. is recovered, due to the solution going to waste in the slimes. To neutralize the acidity of the ore and to aid the settling of the slimes, lime is suspended in a wire basket in the upper part of the tank. The cost of treatment is given as 1 dollar per ton.

Utah.—Mercur is one of the greatest centres of the cyanide process in the United States, the different plants having a capacity of over 2000 tons per day. The Mercur ore has been described as follows †:—"Silica in a form similar to silicious sinter, or geyserite, characterizes the ore. Cinnabar is most abundant in this rock, and forms beautiful incrustations in the cellular varieties. Wherever found in the district, cinnabar is considered a sure sign of gold. Orpiment and realgar occur in large quantities in some of the ore. There is about as much iron as is usually found in impure limestone and clay. Barite and gypsum occur more or less crystallized: also masses of limestone are found mineralized in rings, the outside assaying from 6 dollars to 8 dollars, and the centre a trace in gold. No free gold is visible in the ore, even with a microscope. One remarkable feature is the absence of silver. The average of the ore milled is kept close to 12 dollars per ton."

At the Mercur mill the ore is delivered by the railroad to an ore-bin 40 ft. long, 20 ft. wide, and 20 ft. deep. It is crushed in a Dodge crusher, from which it passes to a set of Walls corrugated rolls, and is finally trammed to the cyanide vats, after being crushed to 1 in. mesh or less. The vats are 12 ft. 8 in. in diameter, and hold 15 tons. They are made of tank iron, with red-wood bottoms. The filter cloth on the false bottom is burlap, and lasts from four to six weeks. From the tanks the solution is conveyed to a collecting tank, from which it is pumped by Blake single-acting pumps to the precipitating room. The zinc boxes are from 24 to 36 in. wide, 10 to 12 in. deep, and about 20 ft. long.

* Hunter, *Engineering and Mining Journal*, Jan. 19, 1901.

† *Mining and Scientific Press*, May 23, 1896.

Fine crushing is found to be unnecessary, as the ore is very porous, and much of it disintegrates into mud when solutions are applied.

It is interesting to note one change made in Mercur practice. Formerly the strong solution was run through the ore continuously, the surface being kept always covered. Now a series of washes is run through, the solution each time being down below the surface. The extraction has been increased thereby, and much time saved on each vat.

The solution used is from 0.1 per cent. to 0.3 per cent. in strength. It was at one time the practice to estimate the strength of the solution by its action in the zinc boxes, and by its alkaline feel. At the present time more acute methods are practised. Still an instance has come within our notice of an operator determining the strength of his solutions wholly by their odour.

At the Mercur mill the practice was formerly to standardize solutions by adding cyanide to the lower end of the zinc boxes, "the judgment of the operator determining the time and amount."

The zinc slimes were dried in an old retort belonging to the amalgamating mill. The door is closed but not luted, and at about 160° C. the product ignites, producing fumes of a complex nature, causing salivation and a headache. The slimes are finally taken from the retort, and the burning completed on a sheet-iron table. This product is then shipped to a smelter for refining.

At present the residues from a 12 dollar ore assay about 1.75 dollars, giving an extraction of 85 per cent. The average ore value is about 6 dollars per ton. The cost of treatment is itemized as follows: Mining, 35 c. per ton; railroad hauling and milling, 80 c.; cyaniding the ore, 1 dollar 35 c.—total cost per ton, 2 dollars 50 cents. The consumption of cyanide is at present about $\frac{1}{2}$ lb. per ton of ore.

The Golden Gate Cyanide Works.—These are said to be the largest and best equipped in the United States.* They were constructed in 1898, and are built on a hillside with eight levels. In order to get the ore to the top of the works it has to be hoisted on an incline 800 ft. long. The mill is 294 ft. wide, and 420 ft. in length up and down the slope. The difference in elevation from top to bottom is 145 ft. The retaining walls, which are 2 ft. wide at the top, and have a batter of 1 ft. in 12, required over 50,000 cb. yards of rubble masonry. The various floors were constructed by blasting out the side hill. The broken stone thus obtained was used for the retaining walls and filling behind them. The mill is driven by power transmitted electri-

* Excerpt from Bosqui's *Cyanide Process*, p.174.

cally a distance of 35 miles at a tension of 40,000 volts. The loss of energy in transmission is said to be only 5 per cent.

At the works the 40,000 volt 3-phase current is transformed to one of 220 volts of 2-phase. The current is delivered at a contract price of 60 dollars per H.P. The first section of the mill contains the coarse crushers, and in the second are the dryers. In the third section is the fine crushing machinery, which consists of four sets of 26-in. rolls, and three sets of 36-in. Berthelet apparatus are used for sizing. There are six elevators, with a lift of 60 ft. The fourth, fifth, and sixth sections contain the roasting furnaces, which are of Brown's straight-line design, four in number. Those intended for arsenical ores are estimated to have a daily capacity of 75 tons, while those for talcose ores are rated at 150 tons. The ore is stirred by the rabblers once each minute. One man attends to two furnaces. The gases are carried from the furnaces through 6 by 8 ft. flues into the main dust chamber, which connects with a steel stack, 8 ft. in diameter and 85 ft. high, located on the hill above the buildings. The top of this stack is 275 ft. above the lowest level of the building. The leaching department, which constitutes section 7, is 60 by 294 ft. It has two floors, the main floor supporting ten tanks 25 by 50 ft. and 5 ft. deep (presumably rectangular), and three solution tanks 20 ft. in diameter and 12 ft. deep. The tanks are supported by masonry piers. They are charged by hand from cars run on bridges over the tanks. The eighth section of the mill, which is 50 by 70 ft. and two stories in height, is the precipitation department. It contains three precipitation tanks, each 14 ft. in diameter and 8 ft. deep. The tailings from the leaching tanks are discharged into cars which are run to the waste dump. The building is constructed of steel.

South Dakota.—The following details of cyanide plants and practice are extracted from a paper contributed by Mr. Chas. H. Fulton, M.E., to the *Engineering and Mining Journal*, January 4, 1902.

The Homestake Company's 1200-ton Tailing Plant at Lead.—This plant takes the tailings pulp from the Golden Star, Eighty, and Highland stamp mills. The pulp is conveyed to the cyanide plant in a 10 in. cast-iron pipe. On its way it passes through the classifier house, where some of the slimes are separated by means of eight large sheet-iron cone-shaped classifiers. These cones have no upward or rising current, but as the ore pulp is charged by pipe at the centre, the sands settle to the bottom and are discharged, while the slimes overflow at the periphery. The sands are then carried to the cyanide plant, where they are fed to seven additional cones and treated as above described. The slimes overflow.

ing at the periphery are again discarded, while the sands are discharged to four-compartment jigs of the Harz pattern, which act as classifiers. One cone feeds four jigs. The jigging is done on beds of pyrites concentrates, relatively coarse screens being employed, except in the last compartment. The stroke varies in the different compartments from $\frac{1}{8}$ in. to about $\frac{1}{2}$ in. The last compartment is widened out in order to compensate for the increased amount of discharge water which accompanies the slimes during their carriage from the tanks. The continuous hutch discharge is taken by open launder to the revolving pipe distributors, which fill the leaching tanks, the peripheral overflow of the tanks still carrying off slimes. Lime in the form of an emulsion is added to the pulp launder which takes the jig discharge.

There are fourteen leaching tanks, 45 ft. in diameter, 9 ft. deep, built of California red-wood. The tanks are placed seven in a row. There is one revolving distributor of the Butters type for each row, being moved from tank to tank by overhead track, as required. To get an even overflow at the periphery of the tank, the tops of the staves are grooved to a depth of $\frac{1}{2}$ in., and a soft pine tongue is inserted, projecting from $\frac{1}{2}$ to $\frac{3}{4}$ in. above the top of the staves. This is then easily planed, and can be kept perfectly level. It is also readily renewed.

It is the object to charge just as much slime with the sands as possible, and still leach in a reasonable time. For this purpose the adjustments are made on the jigs and distributors. The value of the pulp as it leaves the stamp mills is from 1.25 dollars to 1.50 dollars per ton. The amount crushed in the mills approximates 2000 tons per day, while the cyanide mill treats about 1200 to 1300 tons per day. The difference, or 700 tons, is discarded as slimes which, while carrying considerable value, cannot be treated in the condition they are in, and must be discarded. The proportion of slimes separated out is about 35 per cent. The sands are leached directly in the filling vats.

The Cyanide Plant of the Wasp No. 2 Mining Company.—This plant on Yellow Creek, near Kirk, is a dry, coarse crushing plant, capacity 90 tons per day. The ore is quartzite, the gold being on the cleavage planes, and in the seams and cracks, so that coarse crushing only is necessary to make it available for solution. The value varies from 3 dollars to 12 dollars per ton, the main bulk averaging about 4 dollars. It is mined very cheaply (85 to 90 cents) by open-cut mining. The mill is built in terraces on a very steep hillside, and the ore is moved through almost entirely by gravity. It is hauled from the mine 750 ft. distant in trains of four cars, and charged into the main bin, from which it passes to a No. 2 D. Gates crusher over a 4 by 8 ft. grizzly having $1\frac{1}{2}$ in.

spaces between bars. The undersize and the crushed ore from the Gates go to the storage bin, from which it is fed by Tulloch feeders into one set of coarse Gates rolls 14 by 24 in., 80 revolutions per minute. The rolls discharge to a stationary inclined 2-mesh screen 7 ft. long, 1 ft. wide, the oversize from which passes through the finishing rolls 14 by 24 in., operated at 100 revolutions per minute, and meets the undersize, the combined product being elevated by the one-bucket elevator in the mill to the shaking finishing screens, situated on a level with the coarse rolls. This screen is an inclined shaking $2\frac{1}{2}$ -mesh screen, 16 ft. long and 2 ft. wide, the lower half being stationary. Nearly all of the finished product passes a 6-mesh screen, the oversize from which is returned by gravity to the finishing rolls.

The ore is charged from the finished product bin into the four leaching tanks of Oregon fir; each tank is 16 ft. diameter, 7 ft. deep, and of 55 tons capacity. For this purpose a 16 in. belt conveyor is used at a speed of 600 ft. per minute. A large 100-ton tank has recently been added, which gives the mill the capacity mentioned above. Tanks are charged in from two to two and a half hours, the cyanide solution being added after the tank is about one-half filled. The strength of the strong solution is 6 lbs. cyanide per ton of solution. About 15 lbs. of strong solution are run on, and usually allowed to stand three to four hours, then it is drained, a weak solution being added at top to replace it. After this complete replacement it is allowed to stand a short time and then circulated for the balance of the leaching. The weak solution contains from $2\frac{1}{2}$ to 3 lbs. cyanide per ton. About 50 tons of weak solution are used, followed by from 6 to 9 tons of wash water. To neutralize the acidity of the ore and give a protective alkalinity, 6 lbs. of lime per ton are added before the ore is dumped into the first bins. The cost of cyaniding is from 85 to 90 cents per ton. The precipitation is accomplished by zinc thread in specially constructed barrels, and very good results are obtained, the tailings solution having a value of from 4 to 10 cents per ton. The extraction made is from 80 to 90 per cent. of the values, and the consumption of the cyanide amounts to about $\frac{1}{2}$ lb. per ton of ore. The precipitates are treated by the usual sulphuric acid method. Zinc dust precipitation in the tanks, pumping the resultant mixture through filter-presses, was tried at the mill, but discarded on account of the difficulty in the clean-up.

BRITISH COLUMBIA.

Athabasca Mine, Nelson.—In a paper read before the Mexican meeting of the American Institute of Mining Engineers in 1901, Mr. E. Nelson Fell gives some interesting information on the treatment of tailings at this mine. The ore, he says, consists of a quartz gangue, containing a little lime and variable quantities of the sulphides of iron, lead, and zinc. The following figures, giving the analyses of the ore before milling, and of the tailings after milling, which constituted the material to be cyanided, are based on the daily samples taken during February 1901.

Analyses of Ore and Tailings.

				FEBRUARY.	
				Ore.	Tailings.
				Per cent.	Per cent.
Zn,	.	.	.	1.93	0.91
Fe,	.	.	.	7.04	2.65
Pb,	.	.	.	1.63	0.20
CaO,	.	.	.	4.97	2.46
S,	.	.	.	5.99	1.71
Al ₂ O ₃ ,	.	.	.	3.20	3.20
SiO ₂ ,	.	.	.	74.20	85.00
				Oz. per ton.	Oz. per ton.
Au,	.	.	.	1.68	0.32
Ag,	.	.	.	1.32	0.38

The analysis shows this to be an ore well adapted for cyanide treatment.

The plant was located to receive the tailings direct from the mill, in two distributing-tanks, 14 ft. in diameter and 10 ft. in height. The tanks are fitted with annular launders around the rim, and are filled with water before the admission of the tailings; while the overflow is carried off in annular launders.

A slimes-arrester is fixed in each vat, consisting of a sheet of iron, 10 in. wide, fitted inside each vat, about 1 in. from the staves, and extending all the way round. This ring of sheet iron is thus 2 in. less in diameter than the diameter of the vat, and is held in position by eight iron brackets so arranged that the ring can be raised entirely above the level of the tank, or lowered until the upper edge is about level with the top of the staves, according to the nature of the ore being treated. The obvious intention of using this contrivance is to retain as much of the slimes with the sands as possible.

The slimes-arrester described by Mr. Fell is exactly the same as that attached to the J. C. Fraser and Price's continuous grinding and amalgamating pans, which were extensively used in Australia and New Zealand for the treatment of mill tailings before the introduction of the cyanide process.

The overflow takes place in the space between the ring and the staves, and having such a great length of outflow, is very even and quiet.

The plant, which has a capacity of 50 tons per day, consists of five leaching-vats, each 18 ft. in diameter and 4 ft. deep, fitted with centre discharge doors, and two collecting-vats, each provided with the ordinary revolving reaction distributors. The latter are 14 ft. in diameter and 10 ft. deep, and so situated above the leaching-vats, that each one can be discharged by shovelling through side-doors into any one of three adjacent leaching-vats.

Besides these there is a strong gold tank, a weak gold tank, a waste-water vacuum tank, and two series of zinc extractors, each consisting of twelve iron boxes, which are square and have each a capacity of one cubic foot of zinc-shavings. The boxes are movable, and each is independent of the other. The two gold tanks are 10 ft. in diameter and 6 ft. deep, and the two sumps 12 ft. in diameter and the same depth.

Mr. Fell supplies the following working details:—

"This description shows the actual details of treatment, and the results obtained from the treatment of Lot 59, which gives a fair example of the treatment.

"June 26th, at noon, turned on strong solution (0.24 per cent.) until same stood 6 in. deep on the sands; allowed to stand 4 hours. At 4 P.M. opened outlet-cock and allowed solution to run into gold-tanks; at 4.30 P.M., as the last of the solution was passing off, took sample, which assayed "nothing" in cyanide and 9.30 dollars in gold. Closed outlet-cock, admitted fresh charge of strong solution, and allowed to stand 8 hours. At 12.30 (midnight), June 27th, opened outlet-cock, and at 1 A.M. took sample, as before, which assayed 0.06 cyanide and 28.94 dollars in gold. At 1 A.M. turned on weak solution (0.08 cyanide), allowing same to drain through without interruption till 4 A.M. Sample at 4 A.M. assayed 0.10 cyanide and 8.08 dollars in gold. Shut off outlet-cock and allowed solution to stand until 1.30 P.M. Opened outlet-cock and admitted fresh solution (0.06 cyanide) and allowed same to run through, admitting fresh solution as required till 12.30 (midnight), June 28th. Sample taken at 10 P.M. assayed 0.07 cyanide and 0.62 dollar in gold. Closed outlet-cock and allowed to stand until 5 A.M. Opened outlet-cock at 5 A.M. and allowed fresh solution to run through until 11 A.M. Sample

taken at 6 A.M. ran 0·06 cyanide and 0·42 dollar in gold; and sample taken at 11 A.M. ran 0·6 cyanide and 0·21 dollar in gold. At 11 A.M. turned in water-wash till 2 P.M. Sample taken at 12.30 P.M. ran 0·06 cyanide and 0·21 dollar in gold; and sample taken at 2 P.M. ran 0·06 cyanide and 0·21 dollar in gold. At this point the process was declared finished; the wash was drained to waste; and the tailings were discharged.

"Assay of the tailings before treatment gave 13·02 dollars, and after treatment 2·07 dollars in gold. Percentage of recovery, 84·1. Time occupied, 2 days 2·5 hours."

If the actual recovery is equal to the extraction obtained by the assay difference, the results obtained must be considered very satisfactory, and with prolonged leaching could doubtless be increased.

THE CYANIDE PROCESS IN INDIA.

Laurence Pitblado states* that in the Kolar field there are at present six cyanide works in operation. The ores of the field are very simple, consisting mainly of pure quartz, with only a small percentage of iron pyrites. The material treated is tailings from the stamp mills. Those first worked in the Mysore plant averaged 4·5 dwt. gold, and yielded 65 per cent., with an average consumption of 1 lb. cyanide per ton. In 1897 a month's test with 40-mesh screens in the batteries gave the following result:—90·65 per cent. amalgamated in batteries and on plates; 74 per cent. of the value in the tailings recovered by cyanide lixiviation, making a total extraction of 97 per cent. of the ore as delivered to the mill. The cost of cyaniding at the present small plant of the Mysore Company is £2, 10s. 5d. per ton, exclusive of royalty and depreciation, but in the new 4000-ton plant which is being erected alongside the heap of accumulated tailings that is to be worked, it is believed that the cost will not greatly exceed 2s. At the Champion Reefs mill, where 20-mesh screens are used in the batteries, the extraction from the tailings is about 56 per cent. with a consumption of 1 lb. of cyanide per ton.

In refining the precipitate in this district it is first passed through a 30-mesh screen, drained, dried, roasted with or without a small percentage of nitre, and fused directly in plumbago crucibles. At the Mysore works the precipitate is retorted before roasting, yielding about 100 lbs. of mercury per month. The presence of mercury in the zinc boxes generally leads to the production of much-floured and brittle zinc. In fluxing and smelting

* *Journal of the Society of Chemical Industry*, Feb. 28, 1898.

directly, the retorted slimes the bullion assayed 56·4 per cent. gold, 3 per cent. silver, 2·4 per cent. lead, 19·6 per cent. copper, 18 per cent. zinc, and 0·1 per cent. nickel. The slag contained a good many shots of metal. In roasting with nitre a slag freer than the above was produced, and a bullion assayed 49·5 per cent. gold, 4·6 per cent. silver, 4·29 lead, 38·21 per cent. copper, 2·10 zinc, and 0·22 per cent. nickel. These results being unsatisfactory, the following practice was adopted at the Mysore works. The retorted and dried slimes are mixed with 10 per cent. of nitre and roasted at a bright red heat. When cold they are boiled with dilute sulphuric acid (1 : 2), which dissolves the copper. The dried and washed product is fluxed with about 35 per cent. borax, 15 per cent. soda, and 10 per cent. sand, giving a slag free from shots of metal and a bullion assaying 81·3 per cent. gold, 6·9 per cent. silver, 2·71 per cent. lead, 6·78 per cent. copper, 0·4 per cent. zinc, 0·12 per cent. nickel.

CHAPTER XII.

THE SIEMENS-HALSKE PROCESS.

THE distinguishing features of this process are the use of extremely dilute solutions of cyanide and the electrical precipitation of the gold.

Since the introduction of the cyanide process, the precipitation of the gold by metallic zinc has always been regarded as a weak point; and metallurgists have devoted much time in the endeavour to discover an efficient substitute for it.

Electrical precipitation naturally engaged the attention of many investigators. In 1893, the author, assisted by Mr. F. B. Allen, M.A., B.Sc., conducted a number of experiments with electrical precipitation to determine the method of precipitation to be adopted at the School of Mines cyanide plant. Many different modifications were tried. With some, the precipitation from solutions of ordinary working strength was very satisfactory; but, with all, the precipitation of the gold from dilute solutions, such as those corresponding to weak cyanide washes, was always very imperfect and accompanied by decomposition of the water.

In the Siemens-Halske process this difficulty is overcome by causing a slow artificial circulation of the cyanide solutions in the extractor.

The plant and operations connected with the leaching of the gold are the same as those described in the preceding chapters, the only difference being in the extractor-house.

The electrical precipitation of gold has been introduced with marked success at a number of cyanide plants at the Witwatersrand Goldfields, and its use is extending. Up to the present it has not been introduced to New Zealand or Australia, and so far very little has been written about it. For the following details of the process I am indebted to the papers of Mr. Charles Butters and Mr. A. Von Gernet, read before the Chemical and Metallurgical Society of South Africa, and published in the *South African Mining Journal*.

Discovery of the Process.—Mr. Von Gernet said the electrical precipitation of gold extracted from ores by cyanide has been in use in Europe and Asia as far back as 1888. In 1887, Dr. Siemens found that the gold anodes used in electroplating at his works in Berlin lost weight when standing idle in the cyanide solution, without any electric current passing through the bath. This, in connection with the well-known fact that gold was soluble in aqueous solutions of cyanide, first induced him to try the use of that solvent for the extraction of gold from ores.

In the same year he built a small plant to make experiments on concentrates produced in Siebenburgen. The gold was precipitated both by electrolysis and zinc filings. It was found, however, that the zinc method gave good results only from comparatively strong solutions, while the electrical precipitation was effected with both dilute and strong solutions, and its efficiency was not affected by the presence of caustic soda.

Dr. Siemens therefore decided to use electrolysis only, and early in 1888 he commenced operations on a large scale. Engineers were sent to different countries, two going to Hungary, one to America, and one (Mr. Von Gernet) to Siberia.

The operations were generally successful, and in May 1894, a plant, capable of treating 3000 tons of tailings per month, was erected at the Worcester mine, near Johannesburg. During 1895 the process was adopted by some eight or ten large mining companies, including the Metropolitan, May Consolidated, Ceresus G.M. Co., No. 4 Central Works, and Robinson Slime Works; and already it is a formidable rival of the McArthur-Forrest zinc-precipitation process.

Action of the Electric Current on the Cyanide Solution of Gold.—The electric current decomposes the auro-potassic solution, depositing the gold on the negative pole and liberating the metalloid at the positive pole. In a fixed time a given electric current will deposit a certain quantity of metal, which quantity will vary for different metals in direct proportion to their electro-chemical equivalents. This law holds good only for solutions strong in metal; but with very dilute solutions, as in use in the cyanide process, the current does not find sufficient of the metallic compound present at the electrodes, and consequently decomposition of water also takes place; for this reason, to make the precipitation as efficient as possible, constant diffusion of the solution is required.

The artificial circulation of the solution is most economically and conveniently obtained by allowing a slow but steady flow through the precipitation boxes. It is of the highest importance to give a very large surface to the electrodes, since a more efficient

precipitation is obtained by doubling the number of plates than by increasing the current tenfold.

The Cathode or Negative Electrode.—To obtain a satisfactory cathode, a metal must be used which will fulfil the following conditions:—

1. The precipitated gold must adhere to it.
2. It must be capable of being rolled out into very thin sheets to save unnecessary expense.
3. It must be easy to recover the gold from it.
4. It must not be more electro-positive than the anode, in order to prevent return currents being generated when the depositing current is stopped.

The most suitable metal was found to be lead, which, in the form of foil, meets all the requirements, and is therefore used in the Siemens-Halske process.

The Anode or Positive Electrode.—The requirements of the anode are no less important. By the action of the current a metalloid is liberated at the positive electrode, and the latter, when a metal, begins to oxidize. Carbon could be used, but it will not withstand the action of the current, soon crumbling into a powder which decomposes potassium cyanide. Besides, when this finely-divided carbon is in suspension, it cannot be removed from the solution by filtration.

When zinc is used as an anode, it forms a white precipitate of ferro-cyanide of zinc by the reaction of zinc oxide upon ferro-cyanide formed during leaching. In the same way, iron anodes form Prussian blue by the reaction of oxide of iron and ferro-cyanide. In consequence of this reaction, the amount of ferro-cyanide in the cyanide solution does not increase.

The cyanide can be recovered from the Prussian blue, formed at the iron anodes, by dissolving it in caustic soda, then evaporating the solution, and finally smelting with potassium carbonate. Mr. Von Gernet states that this process has been tried on a small scale, about 50 lbs. at a time, with the result that a nice clean potassium cyanide was obtained. In the treatment of clean tailings, this regeneration of cyanide is not of great importance; but with concentrates, or tailings, which decomposes the cyanide solutions with formation of ferro-cyanide, it will effect a considerable saving.

Electric Current Required for Precipitation.—Only a very weak current is required to precipitate the gold from cyanide solutions, a density of 0.05 ampère per square foot being sufficient. With cathodes $1\frac{1}{2}$ in. apart, 7 volt is sufficient to produce this strength of current.

The advantages gained by using such a weak current are:—

1. The gold is deposited hard on the lead-foil.
2. The iron anodes are preserved for a long time, as their waste is in proportion to their current strength. In a plant treating 3000 tons per month, 1080 lbs. of iron are destroyed in that period.
3. Little power is required. 746 Watts equal 1-horse power. A 3000-ton plant requires 2400 Watts, equal, theoretically, to $3\frac{1}{2}$ -horse power, and actually requiring about 5-horse power.

The Advantages of Electric Precipitation.—The principal advantages claimed for this process are as follows:—

1. That the precipitation operates independently of the amount of cyanide or caustic soda present in the solution. Therefore, in the treatment of tailings, very dilute solutions can be used, the only limit being a sufficient quantity of cyanide to dissolve the gold satisfactorily.

A solution containing 0.03 per cent. of cyanide will dissolve gold as effectively as a solution containing 0.3 per cent., provided a longer time is allowed for treatment. In the first case, the decomposition of cyanide is much less than in the second, resulting in a corresponding economy.

2. However acid the solution may be when entering the extractor, the precipitation takes place equally as well as it does when the solution is neutral or alkaline.
3. No complications arise from the formation of lime, alumina, or hydrate of iron, which sometimes cause trouble in the zinc process of precipitation.
4. With ores or tailings containing copper, the extraction of the gold will be the same, but the decomposition of cyanide less than when using stronger solutions.
5. The successful treatment of slimes.

The Actual Working of the Process.—The first practical demonstration of this process on a commercial scale took place at the cyanide works of the Worcester Gold Mining Company, near Johannesburg, under the supervision of Mr. A. Von Gernet.

The plant consists of five leaching vats placed on a row of stone piers, with a single tunnel beneath. Each vat is 20 ft. in diameter, with 10 ft. staves, and has a capacity of 100 tons of tailings.

Between the vats and the electric extractors there are placed two tanks, 16 ft. in diameter, with 6 ft. staves, forming two intermediate reservoirs, which enable the flow through the pre-

cipitation boxes or extractors to be kept constant and steady, a matter of great importance.

A better method to secure an even flow is to pump all the solution into a small raised tank, provided with an overflow into the intermediate tank and a delivery pipe to the precipitation boxes. The small tank is always kept full to overflowing, so that it delivers under a constant hydraulic head.

Beyond the precipitation boxes there are two sumps, 20 ft. in diameter and 6 ft. deep, from which the cyanide solutions are returned to the leaching vats.

Two collecting vats, 20 ft. in diameter and 8 ft. deep, receive the tailings from the 25-stamp battery.

The Electric Precipitation Boxes.—There are four precipitation boxes, constructed of wood, each 18 ft. long., 7 ft. wide, and 4 ft. deep. Each box contains 89 iron-plate anodes, 7 ft. by 3 ft. by $\frac{1}{8}$ in., cased in canvas to retain the small quantity of Prussian blue produced; and 88 cathodes of lead-foil stretched on iron wires fixed on a wooden frame. Each frame contains three strips, 3 ft. by 2 ft., so that, counting the double surface of each lead-sheet, there are altogether about 3000 square feet of cathode surface, the current density being 0.05 ampère per square foot. Copper wires are fixed along the top of the sides of the boxes, and convey the current from the dynamo to the electrodes.

The boxes are made of 3 in. material throughout, with stiffening pieces across the sides and bottom. The divisions are of wood, or are formed by raising some of the iron plates about an inch above the level of the solution, while others rest right down on the bottom, the joints being made water-tight by means of wooden fillets caulked with hemp packing. By this means a series of compartments is obtained, similar to those in a zinc precipitation box, the difference being that the solution passes alternately up and down through successive compartments. The rate of flow is about one foot per minute.

The Clean-up.—The boxes are kept locked, being opened once a month for the "clean-up," which is conducted as follows:—The frames are taken out singly, and the lead-foil is removed and replaced by fresh lead-foil, the whole operation taking but a few minutes for each frame. The lead, which contains from 2 to 12 per cent. of gold, is then smelted into bars and cupelled.

The gold is deposited on the lead sheets as a thin bright yellow film, which adheres firmly to the lead. The consumption of lead at the Worcester Works is 750 lbs. per month, equal to $1\frac{1}{2}$ d. per ton of tailings.

The working expenses for treating 3000 tons per month were as follows:—

	<i>d.</i>
Filling and discharging leaching vats, ...	10·00 per ton.
Cyanide, $\frac{1}{4}$ lb., ...	6·00 „
Lime, ...	1·20 „
Caustic soda, ...	0·50 „
Lead, ...	1·10 „
Iron, ...	2·20 „
White labour, ...	5·20 „
Native labour and food, ...	1·90 „
Coal, ...	4·60 „
Stores and general charges, ...	3·30 „
Total, ...	36·00 per ton of 2000 lbs.

The cost of treatment per ton of 2·240 lbs. would be 3s. 4·32d.

The tailings assayed from 6 dwts. to 8 dwts. of gold, and the residues, after treatment, from 1 dwt. to 2 dwts. per ton. The average actual extraction was about 74 per cent.

The solutions, after leaving the precipitation boxes, still contained gold, the strong solution showing by assay 4 dwts. 8 grs., and the weak solution 10 grs. per ton of solution. On the average, the strong solutions carried from 4 dwts. to 5 dwts., and the weak from 10 grs. to 1 dwt. of gold per ton of solution.

From November 1894 to May 1895, the Metropolitan Company treated 26,900 tons of tailings for 4845 oza. of gold, at a cost of 2s. 8d. per ton. At the May Consolidated the working expenses amounted to about 2s. 4d. per ton, excluding the royalty for the use of the process, which amounts to 3 per cent. The extraction amounted to over 80 per cent. of the original assay value.

Details of the Treatment.—The time occupied in leaching and washing, together with the quantity of the solutions, are given in the following tabulated statement :—

	HOURS.
Alkaline wash, 10 tons, ...	3
Strong cyanide solution, 70 tons, 0·05 to 0·08 per cent.	
KCy, applied in 14 separate portions of 5 tons each,	65
Weak cyanide solution, 21 tons, 0·01 per cent. KCy,	
applied in 3 portions of 7 tons each, ...	18
Water washes, total 11 tons, pumping dry and dis-	
charging, ...	22
Total, ...	108

The working of this process gives rise to the production of a number of valuable commercial bye-products, including copper, lead, litharge, and paint.

CHAPTER XIII.

OTHER CYANIDE PROCESSES.

The Diehl Process.—This is an adaptation of the Sulman-Teed process. It embraces the following essential stages * :—

1. Crushing and sliming the ore.
2. Treating the slimes in agitators with a solution of potassium cyanide in combination with cyanogen bromide.
3. Filter-pressing the sludge.
4. Precipitating the gold from solution by means of zinc.

According to the character of the gold and its associates, amalgamation and concentration can be added to the process.

The most direct advantage of this process is, that the ore is treated in a "raw" condition.

At Hannan Star mill, where the process was first introduced, the ore is dry-crushed in two No. 5 Krupp ball-mills provided with 30-mesh screens. The crushed ore is elevated, mixed with water, and classified into sands and slimes. The sands are conducted to copper-plates on which from 10 to 15 per cent. of the gold is saved. From the plates the pulp travels by gravitation to the fine-milling department, where it is ground into slimes in a large Krupp tube or flint mill, which is an 18 ft. long steel cylinder, 4 ft. in diameter, charged with 4 tons of flint balls. The sand is fed into one end and issues at the other of such a fineness that the whole product will pass through a sieve with 200 mesh per lineal inch.

From the grinding mill the pulp is led to settling vats, from which the surplus water is returned to the mixing machine. The thickened pulp is now led into agitators, where it is treated with a solution of cyanide of potassium and bromide of potassium.

When the agitator has received its full charge, a strong solution of potassium cyanide is added. For slimes containing 1 to 3 oz. of gold per ton, we have found it sufficient to add so much cyanide that there will be 4.4 lbs. KCy per ton of dry material. After the

* Knutsen, *Trans. Inst. Min. and Met.*, London, 1902.

sludge has been agitated for 1 to $1\frac{1}{2}$ hours, the solution of bromide of cyanogen is allowed to flow in, the quantity added being 1·1 lb. per ton of dry material. The agitator is kept going for 24 hours from the time the KCy solution was charged into it. In case the sludge should contain more than 3 oz. per ton, it may be advisable to add, after 6 to 8 hours' agitation, a further quantity of KCy and BrCy, to ensure a good extraction. On the other hand, if the sludge contains less gold than 1 oz. per ton, the quantity of KCy and BrCy can be considerably reduced.

About 2 hours before the agitator is ready to discharge to the filter-press, lime is added to the sludge in quantity varying from 1 lb. to 4 lbs. per ton of dry slimes. In the most cases, I think, 3 to 4 lbs. is used. A cleaner precipitate is thereby obtained in the zinc boxes.

After agitation, the pulp is pressed in Dehne's filter-presses, in which the dissolved gold is washed out. After washing, the dry cakes of slime are thrown out on the dump.

The gold in the solutions is precipitated with zinc turnings in the ordinary way. It is claimed that an extraction of over 93 per cent. can be obtained. The consumption of water is very small in this process, which is an important factor on a goldfield, where water is so scarce. All the available water is salt or brackish, which doubtless tends to retard close concentration.

Mr. Feldtmann gives the following summary of the costs for the month of July 1901, when 2210 tons of ore were treated at Hannan's Brown Hill, at Kalgoorlie, by this process:—

Summary.

	Per ton.	
	<i>s.</i>	<i>d.</i>
Milling, ...	4	0·59
Concentration, ...	1	7·19
Treatment of concentrates, ...	1	4·08
Extraction, ...	17	1·02
	24	0·88

The Schilz Process.—This process is based on the addition of barium peroxide, BaO_2 , to the ordinary cyanide solution, which it is claimed under certain conditions parts with one-half of its oxygen. It is further claimed that the normal oxide of barium, which is left after the treatment, performs further functions—inasmuch as it decomposes the sulphate of iron in the solution, forming sulphate of barium and oxide of iron, both insoluble; it removes sulphocyanides, and it dispenses with the use of lime and

answers the purpose of the same, with other minor advantages in addition. On the other hand, the process requires much longer time to effect a satisfactory extraction than in ordinary cyaniding, thus necessitating a considerable increase in vat capacity.

Herr Schilz states that to ensure success the peroxide must be well sprinkled with the tailings whilst the vat is being filled, so that a good mixture may be obtained. A rather stronger cyanide solution than usual is run in, and the whole allowed to stand undisturbed for at least three days. The longer the time the better is the extraction, and in the case of concentrates it should be at least a week. After running off the gold solution the residue is washed with four weak cyanide solutions, the first of which should remain six hours. Then, again, a strong solution should be applied, followed by weak solutions.

The quantity of peroxide depends less on the percentage of gold than on the length of time during which the solution remains in contact with the charge; and, further, the more pyrites present the greater the consumption of peroxide. The vat should not be filled to the brim with the tailings, as the mass swells up by the evolution of gas, sometimes as much as a foot. Thus the liberated oxygen should remain undisturbed as long as possible in contact with the charge, the greatest solvent action of the cyanide being after a few days. The quantity of peroxide required varies with the material under treatment.

In actual practice the quantity varies from $\frac{1}{2}$ lb. to 1 lb. per ton of ore treated. Delivered on the Rand, the cost of barium peroxide is £50 per ton.

A working trial of the process at the City and Suburban mine, according to the report of the manager, showed a gain of about 13 per cent. in extraction at a small additional cost—that is, a rise from 76 to 89 per cent. The treatment of the pyrites concentrates gave also satisfactory results.

The inventor makes the following claims for his process:—

1. BaO_2 decomposes in contact with moistened tailings into BaO and O , and supplies the cyanide of potassium with oxygen required for the dissolution of gold. This decomposition is performed in the cold.

2. BaO_2 supplies the oxygen which is required for the decomposition and dissolution of the reduced pyrites, and thus assists in liberating the enclosed gold.

3. BaO_2 in decomposing supplies BaO , which, being a strong basis, exerts a purifying and clearing influence upon the cyanide solutions, thus increasing their solving power.

4. BaO_2 renders superfluous the use of lime, the now existing use of which has in its train so many injurious secondary effects.

The Park-Whitaker Cyanide Process.—This process is intended for the treatment of cupriferous ores and concentrates which cannot be treated successfully by the ordinary cyanide processes, on account of the solubility of copper ores in cyanide solutions.

In this process the ore is subjected to a chloridizing roasting, after which the soluble copper chlorides are removed by leaching with water. An alkaline wash is then applied, and the gold and silver extracted with a dilute solution of cyanide.

During the roasting the silver sulphides present are converted into chloride, which is readily dissolved by cyanide. The dissolved copper is recovered by passing the solutions through iron turnings or scrap-iron.

Experiments on a working scale were made by the author on a parcel of ore from the Monowai mine, N.Z., with most successful results, and preparations are now being made for more extensive trials.

The Pneumatic Cyanide Process.—In this process the dissolution of the gold is accelerated partly by jets of compressed air, which cause a continuous and gentle agitation of the pulp, and partly by the aeration of the solution caused by the passage of the air through the charge.

The compressed air is conducted to the charge through a coil of perforated pipe placed on top of the filter-cloth.

This process is identical with the Park-Horn process, for which provisional letters patent were obtained in New Zealand in 1895. Tests were made by Mr. George Horn and the author at Kuaotunu goldfield on excessively fine slimes. The dissolution of the gold was rapid and almost complete, but the mechanical difficulty of separating the gold-containing solutions from the slimes gave so much trouble that further attempts were abandoned. For the treatment of sands, ordinary percolation was found to give satisfactory results without artificial aeration.

Gilmour-Young Process.—This process has been operated on argentiferous gold ores in Central America. It is essentially an amalgamation process, closely following the Washoe method of treatment. The addition of a caustic alkali, copper and zinc amalgam, and mercury to a pulp containing cyanide solution must promote many complicated reactions of doubtful utility.

ELECTRO-CHEMICAL SLIME PROCESSES.

These are based on the dissolution of the gold with cyanide, and the electro-precipitations of the gold on mercury, which also serves to amalgamate any gold too coarse to be dissolved by cyanide.

The Riecken Process.—This process has lately been installed at the South Kalgoorlie mine, Kalgoorlie, and since December 1900 is reported to have treated 5000 tons of slimes. At present the capacity of the plant is being doubled. The following details of the process are extracted from a description supplied to the *Australian Mining Standard* in 1901.

In this process the pulverized and, if necessary, roasted ore is agitated in a vat of special construction with cyanide solution; a current of electricity is passed through the resulting pulp, by which the gold dissolved in the cyanide solution is deposited on the amalgamated sides of the vat as amalgam, while at the same time any coarse particles of gold in the ore too large to be dissolved by the solvent are mechanically amalgamated. Thus, in one operation, in a single vat, are simultaneously performed—(1) amalgamation of the coarse gold; (2) solution by cyanide of the fine gold; (3) electrical precipitation of this dissolved gold. The pulp, being thus deprived of its precious metals, is at once discharged from the vat to residues dams, and the treatment is finished.

The apparatus in which this operation is performed may be regarded as a huge electro-depositing cell, which is an iron vat with vertical ends, inclined sides, and a rounded bottom. Its dimensions are 13 ft. 4 in. long, 8 ft. 3 in. wide at the top, and 11 ft. deep. It holds a charge of 18 tons. Agitation of the pulp is effected by paddles attached to a horizontal shaft working through stuffing boxes. The paddles are 2 ft. 8 in. long, 5 in. wide, and spaced 12 in. from centres. The ends of paddles reach about 2 in. from the bottom of vat.

The sides and bottom of vat are lined with movable amalgamated copper-plates, $\frac{1}{16}$ in. thick, which form the cathodes or negative poles.

When in operation a thin sheet of mercury constantly descends over the copper-plates from a $\frac{3}{4}$ in. pipe perforated with $\frac{1}{16}$ in. holes, spaced 6 in. centres. A slow reciprocating motion is imparted to these perforated iron pipes, causing them to swing through a chord of 6 in. about twenty times per minute. The mercury finds its way to the bottom of the vat, whence it is drawn off by a trap and elevated by an air-lift to the upper receptacle, whence it again flows into the vat, thus ensuring a continuous circulation.

The anodes are bars of iron 1 in. thick by 3 in. wide, suspended from two girders and curved so as to be parallel to the cathodes. There are twelve cathodes in the vat.

The agitator running about twelve revolutions per minute is found sufficient to keep the pulp from settling. The salt in the local water renders the conductivity of the pulp excellent, so that a current of very low potential is quite sufficient to effect the precipitation of the dissolved gold. The current required for each vat of 18 tons is about 250 ampères of a potential of 2.5 volts, equal to about six-sevenths of an electrical horse-power, the cost of which per ton is merely nominal.

The ore is first pulverized until all the gold is liberated from its matrix. If necessary the ore is roasted. In order to obtain good results it is merely necessary that the gold should be amalgamable, or in such a fine state as will permit of dissolution by cyanide solutions. Necessarily each particular ore requires a certain specific preliminary preparation, determinable by experiment, to obtain the best results.

The pulp, containing approximately equal weights of ore and solution, is discharged into the vat previously described, and which is called the "electro vat." The agitator is set in motion, the electric current from the dynamo is started, and the flow of quicksilver is maintained by means of the air-jet, as already described. The solution is made up of potassium cyanide to the average strength of 0.075 per cent. With a high-grade ore of 2 oz. or over the above-described operation is continued for about eighteen hours, while with low-grade slimes, carrying but 3 dwt. or 4 dwt., it may be complete in about six hours. After a proper time has elapsed the discharge-valve at the bottom of the vat is opened and the pulp is allowed to flow to the residues dam; the vat is refilled as before, and the operation indefinitely repeated. If water is scarce, the supernatant liquor, after the pulp has settled in the dam, may be pumped back into the mill to make pulp for succeeding charges.

The clean-up is effected by withdrawing one copper plate at the time, and replacing it with a spare one. The amalgam produced is very fine, and contains about 27 per cent. of fine gold. In this process sodium is added to the mercury automatically to keep it active and bright.

The Riecken process dispenses with percolation, filter-pressing, or decantation for the removal of the dissolved gold from the ore, and entirely does away with zinc-precipitation and clean-up, the complete treatment of the ore, after its mechanical preparation, being effected in one operation in one vat, in the minimum of time, the gold being recovered as amalgam, requiring only retort-

ing and melting. The future applications of this process will be watched with much interest by metallurgists.

The Keith Electro-Cyanide Process.—The Keith electro-cyanide process is the invention of Dr. Keith, an American electrician. The process consists of two parts. First, the process for dissolving the gold out of the crushed ore; and second, the recovery of the gold from the solution. Dr. Keith's improvement in the dissolving process consists in adding to the solution of potassium cyanide a certain amount of cyanide of mercury. In practice, he states that he finds the best results are obtained when the solvent contains 0.05 per cent. of potassium cyanide and 0.025 per cent. cyanide of mercury. This mixture of cyanides, it is claimed, operates very much faster than the simple potassium cyanide.

The process for the recovery of the gold from the solution is an electrolytic one. The gold and the mercury are deposited together upon amalgamated copper-plates. The amalgam so deposited is scraped off and the gold recovered by distilling off the mercury in the usual way. The anode is not allowed to dip into the cyanide solution, but is placed in a separate compartment and surrounded with a solution of an alkaline salt, so that the cyanide does not become decomposed. The electro-motive force of the current need not be more than half a volt.

CHAPTER XIV.

ANTIDOTES FOR CYANIDE POISONING.

ALL cyanides are deadly poisons ; but the aqueous solutions used in practice are so dilute that there is little or no danger from the prussic acid evolved from them if the buildings are properly ventilated.

Acids react on cyanides, liberating prussic acid gas, which causes almost instant death when inhaled in a pure state. When diluted with air, it causes faintness, dizziness, and a depressing frontal headache.

Even very dilute solutions of cyanide are poisonous when taken internally, and, when they come in contact with the skin, produce, in some persons, an eruption of painful red boils. In cases where the hands and arms must be brought into contact with the solution, rubber gloves, reaching over the elbows, should be provided for the workmen. Kaffir workmen are said to suffer no inconvenience whatever from the contact of their skin with cyanide solutions.

Considering the extensive use of cyanide, the number of fatal accidents is remarkably small. Up to the present time only a few fatal cases have been recorded.

In a cyanide plant, poisoning may be apprehended from two principal sources, namely :—

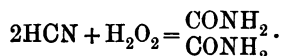
- (a.) From hydrocyanic acid liberated in vat-house.
- (b.) From poisonous gases liberated during acid treatment of slimes.

In South Africa, Australia, and countries where the vats are not covered by roofs, danger from prussic acid vapours, liberated by the action of mineral acids or atmospheric carbonic acid, is practically unknown. Even where the vats are enclosed in a shed, the risk can be reduced to a minimum by the free circulation of fresh air.

The author has observed that the presence of HCy vapours is

always more noticeable in agitation than in percolation plants, the obvious reason being that agitation is generally adopted for the treatment of pyritic ores and concentrates, while the strength of the solutions used is nearly always high.

In cases of cyanide poisoning by inhaling the fumes of hydrocyanic acid, a German chemist recommends the use of hydrogen peroxide, H_2O_2 , which forms with HCy the harmless compound oxamide, the reaction being represented by the following equation:—



This is said to be the most reliable and satisfactory remedy known at the present time.

In a case of HCy poisoning at the Crown Deep, the effect of the gas was immediate, one of the workmen falling as if dead. The same effect was observed at the N.Z. Crown mines, where a foreman fell into a cyanide vat without previous warning and died immediately.

Danger during Acid Treatment of Slimes.—The slimes generally contain a small proportion of insoluble cyanide salts, which yield hydrocyanic acid when the sulphuric acid is poured on them. To guard against this danger, respirators should be used by the workmen who have to stoop over the dissolving tubs,

In cases of poisoning, subcutaneous injections of H_2O_2 are said by Mr. T. L. Carter of the Crown Deep mine to enable the patient to soon come to.

In ores containing arsenic, most of which are more or less soluble in cyanide, there is a danger of arsenic being precipitated with the gold in the zinc boxes. During the acid treatment of such slimes, the deadly poison arseniuretted hydrogen would be liberated by the action of the sulphuric acid on the zinc.

The symptoms observed in the case of the North Pole Company's mill, where the superintendent and foreman both died, while many others were affected, were first nausea, then extreme langour, with pains in the legs, followed by discoloration of the skin in patches assuming the hue of tan; the whites of the eyes became yellow as in jaundice; finally, the passing of blood instead of urine to such an extent that the fluid coagulated in a few hours, the patients apparently dying from internal mortification.

The arsenic being inhaled, passes from the lungs through the whole system, and rapidly attacks the tissues of the body, precluding any relief by means of antidotes.

Where the acid treatment is used, the zinc for precipitation purposes should be free from arsenic; and in all cases the dissolu-

tion of the zinc should be conducted in a special chamber or cupboard connected with a chimney having a good draught.

In cases of internal poisoning, vomiting should be induced at once by emetics or physical means.

Freshly precipitated carbonate of iron, obtained by mixing equal quantities of sodium carbonate and ferrous sulphate, is recommended for internal use.

It was lately reported in the press that Johann Antal, a Hungarian toxicologist, had found that a solution of cobalt nitrate was a perfect antidote for prussic acid poisoning. Recent investigations, however, have shown that cobalt salts exert a toxic action on animals when injected subcutaneously, finally leading to death; and for this reason nitrate of cobalt cannot be recommended for human subjects.

Cyanide Sores.—A percentage of the workmen engaged in the "clean-up" of the zinc extractors are affected with painful sores in those parts of their hands or arms which come in contact with cyanide solutions. Why some men should be exempt and some afflicted in this way is not very clear, but it is probably connected with constitutional causes.

Writing on this subject, Mr. A. Watt supplies the following instructive notes* :—

"These painful affections may arise from two principal causes: first, from dipping the hands or arms into cyanide baths to recover articles which have dropped into them—a very common practice and much to be condemned; and second, from the accidental contact of the fingers or other parts of the hand, on which a recent cut or scratch has been inflicted, with cyanide solutions. In the former case, independent of the constitutional mischief which may arise from the absorption by the skin of the cyanide salts, the caustic liquid acts very freely upon the delicate tissue of the skin, but more especially upon the parts under the finger nails. We have known instances in which purulent matter has formed under the nails of both hands from this cause, necessitating the use of the lancet and poulticing. Again, when cyanide solutions come in contact with recent wounds—even very slight cuts or abrasions of the skin—a troublesome and exceedingly painful sore is sure to result, unless the part be at once soaked in warm water; indeed it is a very good plan, after rinsing the part in cold water, to give it a momentary dip in a weak acid pickle, then soak it for a few moments in warm water, and after wiping the part dry with a clean rag or towel, apply a drop of olive oil and cover up with a strip of thin sheet of gutta-percha."

* Watt, *Electro-Deposition*, p. 611.

Provision of Remedies.—In order to minimize the danger attending cyanide poisoning, the necessary antidotes should be provided in every cyanide plant, and kept in a closed cabinet with a glass front, placed in some conspicuous and easily accessible part of the works, known to all the workmen.

The cabinet should have the words *Antidotes for Cyanide Poisoning* marked in clear letters near the top, and written or printed instructions how to apply the remedies pasted inside the cabinet.

The cabinet should contain the following articles :—

1. A sterilized glass flask marked A, filled with a 3 per cent. solution of hydrogen peroxide. The neck of the flask should be drawn to a fine point and sealed in a blow-pipe flame.
2. A sterilized glass flask marked B, filled with a 30 per cent. solution of hydrogen peroxide, sealed with flame.
3. A hypodermic syringe, made of glass.
4. A stomach tube and funnel.
5. A small conical-shaped medicine glass.
6. A small triangular file.
7. A small pair of pinchers.

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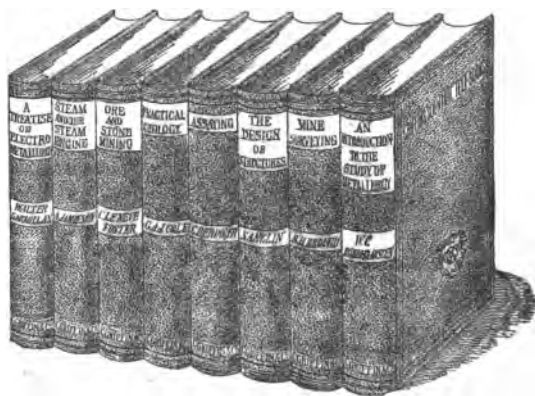
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